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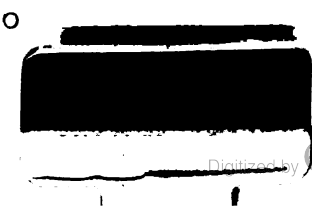

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PREFACE

This volume contains the papers and discussions presented at the New York meeting of February, 1916, excepting those on iron and steel, which were published in Vol. LIII.

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[1] Until Feb., 1917. [2] Until Feb., 1918. [3] Until Feb., 1919. [4] Until Feb., 1920.

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Monday, February 14, 1916

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Wednesday, February 16, 1916

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TECHNICAL SESSIONS

Monday Morning, Feb. 14, 1916.—The opening session, like all the technical sessions, was held at the headquarters of the Institute in the Engineering Societies' Building, New York City. It was under the auspices of the Committee on Petroleum and Gas. Dr. David T. Day, Vice-Chairman of that Committee, presided.

The following papers were presented by their authors or authors' representatives:

Development of the Law Relating to the Use of Gas Compressors in Natural Gas Production. By Samuel S. Wyer.

Necessary Use and Effect of Gas Compressors on Natural-Gas Field Operating Conditions. By Samuel S. Wyer. (Discussed by David T. Day, I. N. Knapp, W. L. Saunders, Harrison Souder, F. G. Clapp.)

The Evolution of Drilling Rigs. By R. B. Woodworth. (Discussed by Chester W. Washburne, I. N. Knapp, W. L. Saunders, Samuel S. Wyer, Leonard Waldo, David T. Day, E. Gybbon Spilsbury.)

The following papers were presented by title:

The Control of Petroleum and Natural Gas Wells. By Alfred G. Heggem. (Discussed by I. N. Knapp.)

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Monday Afternoon, Feb. 14, 1916.—This session was under the auspices of the Committee on Coal and Coke, and S. A. Taylor presided.

The following papers were presented by their authors or authors' representatives:

Illumination of Mines. By Robert P. Burrows. (Discussed by Edwin M. Chance, T. M. Chance, G. S. Rice, D. B. Reger, R. V. Norris, E. T. Lednum, R. Dawson Hall, Hugh Archbald, George H. Stickney.)

Some Researches on Fire-Damp. By Enrique Hauser. (Discussed by George A. Burrell, W. E. Gibbs, E. M. Chance, T. M. Chance.)

The following papers were presented by title:

Economies in a Small Coal Mine. By H. A. Everest. (Written discussion by Newell G. Alford.)

The Effect of Aëration and "Watering Out" on the Sulphur Content of Coke. By J. R. Campbell.

Notes on Brown-Coal Mining in Germany. By George J. Young.

The Stresses in the Mine Roof. By R. Dawson Hall. (Discussed by Charles Enzian, Hugh Archbald, T. M. Chance.)

Monday Afternoon, Feb. 14, 1916.—This session was under the auspices of the Committee on Geology, Professor Alfred C. Lane presiding.

The following papers were presented by their authors or authors' representatives:

Magmatic Differentiation in Effusive Rocks. By Sidney Powers and Alfred C. Lane. (Written discussion by N. L. Bowen.)

The Iron Deposits of Daiquiri, Cuba. By Waldemar Lindgren and Clyde P. Ross. (Discussed by Joseph T. Singewald, Jr., Benjamin Leroy Miller, M. Roesler, L. C. Graton, Harrison Souder, Charles P. Berkey, Alfred C. Lane, J. D. Irving.)

Interpretation of Assay Curves for Drill Holes. By Edward H. Perry and Augustus Locke.

The following papers were presented by title:

The Disseminated Copper Ores of Bingham Canyon, Utah. By J. J. Beeson.

Geology of the Ore Deposits of the Tintic Mining District. By Guy W. Crane.

Observations on Certain Types of Chalcocite and Their Characteristic Etch Patterns. By C. F. Tolman, Jr. (The three papers named above were discussed by L. C. Graton, Alfred C. Lane, J. T. Singewald, Jr., Charles P. Berkey. Written discussion by H. E. Merwin.)

The Iron Ores of the Philippine Islands. By Wallace E. Pratt.

Tuesday Morning, Feb. 15, 1916. Annual Meeting, President W. L. Saunders presiding.

Seventy-two members were present and 1,440 voted by letter ballot.

The minutes of the Annual Meeting of Feb. 16, 1915, were read, and on motion, duly made and seconded, were approved.

Reports of the President,[1] Secretary and Treasurer[2] were presented in writing.

Report of the Tellers for Election of Officers and Directors was presented in writing and the following Officers and Directors were declared elected.

Director, and President,	L. D. Ricketts,
Director and Vice-President,	Karl Eilers,
Director and Vice-President,	James MacNaughton,

[1] This volume, p. xxiii.
[2] This volume, pp. xxv and xxviii.

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Director,	George D. Barron,
Director,	Charles W. Goodale,
Director,	Edwin Ludlow,
Director,	Charles F. Rand,
Director,	Thomas B. Stearns.

The report of the Tellers on the Amendments to the Constitution was presented in writing and the amendment was declared carried by a vote of 1,039 to 401.

The reports of the following committees were presented:

Committee on Membership,
Finance Committee,
Library Committee,
Committee on Papers and Publications.

A petition was presented that the question of the Institute adopting simplified spelling be submitted to letter ballot and was ordered to take the usual course.

On motion, duly made and seconded, telegrams were sent to Dr. James Douglas and Dr. James F. Kemp expressing greetings of the members.

Tuesday Morning, Feb. 15, 1916. Technical Session, President W. L. Saunders presiding.

The following papers were presented by their authors or authors' representatives:

Notes on Flotation. By J. M. Callow. (Discussed by R. H. Richards, J. W. Richards, Leonard Waldo. Written discussion by George D. Van Arsdale.)

Broken Hill Underground Mining Methods. By E. J. Horwood. (Discussed by Albert R. Ledoux.)

The following papers were presented by title:

Grinding Brass Ashes in the Conical Ball Mill. By Arthur F. Taggart and R. W. Young.

Underground Mining Methods of Utah Copper Co. By Thomas S. Carnahan.

Tuesday Afternoon, Feb. 15, 1916.—R. M. Catlin presided.

The following papers were presented by their authors or authors' representatives:

Tests on Motor-Driven Equipment for Use in Preparation of Anthracite Coal. By H. M. Warren, A. S. Biesecker and E. J. Powell. (Discussed by R. V. Norris, William Kent. Written discussion by K. A. Pauly.)

The New Electric Hoist of the North Butte Mining Co. By Franklin Moeller. (Discussed by Graham Bright. Written discussion by K. A. Pauly.)

Conservation of Iron Ores. By C. K. Leith.

The following papers were presented by title:

An Electro-Hydraulic Shovel. By Frank H. Armstrong.

Application of Electric Power to Mining Work in the Witwatersrand Area, South Africa. By J. N. Bulkley. (Discussed by Graham Bright. Written discussion by K. A. Pauly.)

Conservation and Economic Theory. By R. T. Ely.

Pennsylvania Fire Clay. By L. C. Morganroth. (Discussed by David B. Reger.)

The Use of Low-Grade Phosphates. By James A. Barr.

Wednesday Morning, Feb. 16, 1916.—This session was under the auspices of the Committee on Precious and Base Metals, George C. Stone presiding.

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The following papers were presented by their authors or authors' representatives:

The Behavior of Stibnite in an Oxidizing Roast. By H. O. Hofman and John Blatchford. (Discussed by Robert H. Richards.)

The Determination of Grain Size in Metals. By Zay Jeffries, A. H. Kline and E. B. Zimmer. (Discussed by Alfred C. Lane, C. H. Fulton, W. M. Corse.)

Substitutes for Platinum. By F. A. Fahrenwald.

The Newnam Hearth. By William E. Newnam.

The following papers were presented by title:

Determination of Antimony in the Products Obtained by Roasting Stibnite. By W. T. Hall and John Blatchford.

Recent Advances in the Chemistry of the Cyanogen Compounds. By J. E. Clennell.

Recrystallization of Cold-Worked Alpha Brass on Annealing. By C. H. Mathewson and Arthur Phillips. Written discussion by Zay Jeffries.)

Segregation in Gold Bullion. By James H. Hance. (Discussed by Frederic P. Dewey, Francis P. Sinn, E. Gybbon Spilsbury.)

Wednesday Afternoon, Feb. 16, 1916.—This session was under the auspices of the Iron and Steel Committee, J. W. Richards, presiding.

The following papers were presented by their authors or authors' representatives:

Measurement of the Temperature Drop in Blast-Furnace Hot-Blast Mains. By R. J. Wysor. (Discussed by Leonard Waldo, J. W. Richards, Linn Bradley.)

The Control of Chill on Cast Iron. By G. M. Thrasher. (Discussed by Albert Sauveur, Richard Moldenke.)

Magnetic Studies of Mechanical Deformation in Certain Ferro-Magnetic Metals and Alloys. By H. Hanemann and P. D. Merica. (Discussed by J. A. Mathews, Leonard Waldo.)

Effect of Carbon on the Physical Properties of Heat-Treated Carbon Steel. By J. H. Nead. (Discussed by Albert Sauveur, J. A. Mathews, Frank N. Speller. Written discussion by E. D. Campbell.)

The following papers were presented by title:

A Chemical Explanation of the Effect of Oxygen in Strengthening Cast Iron. By W. McA. Johnson. (Discussed by J. W. Richards.)

Manganese-Steel Castings in the Mining Industry. By Walter S. McKee.

Washed Metal. By Henry D. Hibbard and Edward L. Ford.

The Iron Mines of the Sierra Menera District of Spain. By Messers. Sota, Asnar, and Callen.

Wednesday Afternoon, Feb. 16, 1916.—This session was under the auspices of the Iron and Steel Committee, J. W. Richards presiding.

The following papers were presented by their authors or authors' representatives:

Modern Development in the Combustion of Blast-Furnace Gas with Special Reference to the Bradshaw Gas Burner. By K. Huessener. (Discussed by K. Nibecker, S. K. Varnes, J. W. Richards.)

Vacuum Fused Iron with Special Reference to the Effect of Silicon. By T. D. Yensen. (Discussed by J. A. Mathews, W. E. Ruder, J. W. Richards.)

Metallography of Steel for U. S. Naval Ordnance. By H. E. Cook. (Discussed by Albert Sauveur, J. W. Richards, Richard Moldenke, W. E. Ruder, Leonard Waldo.)

The following papers were presented by title:

Iron Ores of California and Possibilities of Smelting. By C. Colcock Jones.

Manufacture and Tests of Silica Coke-Oven Brick. By Kenneth Seaver. (Discussed by J. W. Richards, W. H. Blauvelt, F. G. Bryer, C. G. Atwater.)

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The Duplex Process of Steel Manufacture at the Maryland Steel Works. By F. F. Lines.

The Electric Furnace in the Foundry. By William G. Kranz.

Commercial Production of Sound Homogeneous Steel Ingots and Blooms. By E. Gathmann.

Suggestions Regarding the Determination of the Properties of Steel. By A. N. Mitinsky. (Discussed by Albert Sauveur, J. W. Richards, William Kent, Leonard Waldo.)

Entertainment of Ladies

The complete plans which had been made for the entertainment of ladies at this meeting, and the success of the efforts of the Ladies' Committee during the past few years bore fruit in the attendance of a still greater number of ladies than in any recent year. There was also manifested a most cordial spirit of appreciation on the part of the visiting ladies, as exemplified on Thursday's boat trip by their presentation of a bunch of violets with orchid to each member of the Ladies' Committee, and also the presentation to the Chairman of the Ladies' Committee, of a large bunch of roses with an appreciative letter from one of the visiting ladies. The visiting ladies were met as usual by members of the Ladies' Committee in the Ladies' Reception Room each day from 12:00 to 12:30 o'clock and were then escorted to luncheon. The entertainment for the ladies comprised an Ice Carnival at Hotel Biltmore on Monday afternoon; a visit to the Hippodrome where, besides the usual exhibitions, there was some very fine ice skating, on Tuesday afternoon; a visit to the Art Galleries of Senator William A. Clark on Wednesday afternoon, where refreshments were served, and also tea at the residence of Mrs. Bradley Stoughton. On Wednesday evening there were 120 ladies at the Banquet and nearly this number attended the all-day Boat Excursion on Thursday.

Social Features of the Meeting and Visits

Annual Dinner.—The Annual Dinner was held on Wednesday evening, Feb. 16, at Hotel Astor. The attendance included 358 members and guests and the dinner was followed by dancing. The Dance Committee was composed of:

James T. Kemp, *Chairman*,
de Courcy B. Browne,
Prof. William Campbell,
Richard B. T. Kiliani,
R. V. Norris, Jr.,
Benjamin F. Tillson,
Roger L. Winsley.

College Night.—Many colleges were represented at the meeting of the Institute and graduates who were present met together for a social evening in individual groups numbering from 6 to 60 at their College Clubs or elsewhere, including some dinners at the Engineers' Club, and especially a dinner and smoker for Columbia graduates at the Columbia University Club.

Luncheons.—On each of the three days luncheons were served in buffet fashion in the room adjoining that in which the technical sessions were held.

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New York Club Privileges.—The following New York Clubs very courteously extended their facilities to visiting members during the meeting and honored cards of introduction from the Institute:

Chemists' Club	Machinery Club	Technology Club
Columbia University Club	Princeton Club	Williams Club
Harvard Club	Rocky Mountain Club	Yale Club

Smoker.—The smoker, with its unique and highly entertaining features, was a much appreciated event, regarded as an innovation by those who have attended recent meetings. It was held at the Aldine Club, 200 Fifth Avenue, and was attended by 250 men. The entertainment was entirely by members of the Institute, except for the music. The program was in charge of T. T. Read, as "mine foreman," ably assisted by James T. Kemp and other "shift bosses," undergraduates of Columbia School of Mines. It included an opening address by David H. Browne, a remarkable Talking Doll with T. B. Stearns as interlocutor; the Metallurgical Cow, explained by David H. Browne; original verses by Col. A. M. Hay; pseudo-serious lecture and demonstration of Flotation Process by Gilbert Rigg, assisted by other members, especially Dr. A. R. Ledoux, W. B. McKinlay and Burr A. Robinson; reminiscences by some of the older members, including Dr. R. W. Raymond, E. C. Pechin and Philip N. Moore; and moving pictures interspersed with songs and refreshments. A little booklet of songs was distributed to all through the courtesy of *Metallurgical ond Chemical Engineering.*

Boat Excursion.—Many members and guests of the Institute assembled at the Brooklyn Navy Yard on the morning of Thursday, Feb. 17, and were taken over the Yard by guides provided by Admiral Usher. At 12:00 o'clock sharp 262 members and guests left the Navy Yard on board the S. S. "Dolphin." Luncheon was served on board and the party was landed at Sandy Hook at 2:30 p. m. More than an hour was devoted to an exhibition of the works at Sandy Hook and to watching the firing of some of the guns. The mist prevented the firing of some of the big guns on account of the danger thereof, but the visit very much interested all who took part. The "Dolphin" then brought the party back and landed it at South Ferry. Notwithstanding the season of the year, the temperature did not prove too cold to mar the pleasure of the occasion and those who were in danger of becoming chilled engaged in dancing on deck to the music of the splendid band which had been provided by the authorities. Appreciation was expressed on all sides for the courtesy of the Secretary of the Navy in extending the use of the "Dolphin" through the members of the Naval Consulting Board, who are members of the Institute, and to the Officers of the Navy and the Commandant and Officers at Sandy Hook who did so much to make the trip pleasurable and interesting.

The Daily Fume.—An account of the meeting would not be complete without saying a few words of appreciation of the Daily Fume, which was edited anonymously and issued each day by courtesy of the *Engineering and Mining Journal.* This paper was written in lighter vein and almost boasted in the first issue that accuracy was deliberately sacrificed to interest. The publication of this daily, announced as the official organ of the American Institute of Dining Engineers, did much to enliven the social features of the meeting.

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ADDRESS OF PRESIDENT, W. L. SAUNDERS, ANNUAL MEETING, NEW YORK, FEB. 15, 1916.

The Institute is at present in sound condition professionally and financially. During the past year 546 new members were elected, the total membership now numbering 5,221. The Treasurer's Report shows that the expenses of the Institute are met by its income. Your Directors have thought it wise to issue a referendum to the members asking for a vote upon the suggestion that the annual dues be raised from $10 to $12. This moderate increase is proposed not because the Institute needs it at present but to insure a continuation and increase in its activities and usefulness, and to provide a reserve fund for future emergencies. In well-conducted organizations, such as clubs, it is a wise and safe course to so adjust the dues of members that the total receipts from this source will about equal the annual expenditures, the initiation fees being used for reserve purposes. Our dues alone do not at present meet the expenses. This, I think, is the whole situation, so far as this question is concerned. Expenses might be reduced in some measure, but they cannot be materially reduced without affecting the integrity and usefulness of your Institute.

Nothing of consequence in our internal affairs has occurred during the past year except, perhaps, the establishment of the Arizona Section and the prospect now assured of a Nevada Section.

During recent months there has been a marked change in the relations of the Institute to its fellow societies and to the Government of the United States. That change is one of coöperation. The Secretary of the Navy has appointed two of your members to the Naval Consulting Board to coöperate with members of 10 other scientific organizations. This Board is now organized, and through its work your Institute is brought in close touch with the activities of the other societies and with the United States Navy. The President of the United States has requested us to recommend to the Government one of our representatives in each State in the Union to act in collaboration with a member in each State from the American Society of Civil Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers and the American Chemical Society, thus organizing a directorate of engineers for each State, that will conduct, through the members of each of the five societies referred to, living in each State, a campaign of industrial preparedness. No more important step than this, it seems to me, has been made in the history of your Institute. The work which these 48 Boards are expected to do will be directed by a committee of the Naval Consulting Board, and the far-reaching effect and usefulness of this work can hardly be over-estimated. We hear a great deal about preparedness. Many plans have been suggested for increases in the army and navy of the United States and much difference of opinion

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exists on the subject. As far as I have observed, there is no difference of opinion on the question of industrial preparedness, which means a coördination and coöperation with the Government of our mines, mills, works, and factories, so that in the event of trouble they may be in a position to respond promptly, energetically, and with full force, to the needs of the nation. Even as a peace measure such an organization is desirable. This country is the largest nation in the world industrially, yet our industries are working more or less at cross purposes, are not in touch with each other or with the Government. Such contact as has existed in the past between business and Government is a special, not a general, contact. Certain industries have had the ear of the Government, while others have not. Nor is the Government at present in a position to feel the pulse of the great industrial strength of the United States in peace or war. In this and in many other ways a general contact will not only be beneficial to the Government but should prove of wholesome advantage to our American industries.

The engineer has been called upon to take this important step. Who is better fitted to do it and do it well? Mind you, it is not the mining engineer alone, but the civil, the mechanical, the electrical and the chemical engineers, coöperating as a unit. We have heard a great deal about the desirability of coöperation among engineers, but so far it has been mainly academic. Here we have a practical fulfillment of our desires and an opportunity the importance of which can scarcely be measured. It is no less important to the engineer and to the whole profession he represents than it is to the industrial strength and prosperity of the nation. The engineer is essentially a man of action, an executive, an administrator, not a mere scientific worker, operating behind closed walls in a dusty laboratory or over a drawing board. His place is out in the middle of the road, with his coat off, leading men, and initiating and directing measures of usefulness to the whole people. We have heard our profession defined as one which uncovers the hidden forces of Nature and puts them at the service of mankind. We have done a good deal in the line of uncovering these hidden forces, but have been very slow in our activities in placing the things revealed in useful operation. Modesty is not only a characteristic but a fault among engineers. Some consider it undignified to get what they call notoriety, but it seems to me that there is a difference between notoriety and reputation; the way to get reputation is to do things, not under a bushel, but in the light, so that the world may know and the full measure of benefit may be derived by inspiring public confidence not alone in what has been achieved, but in the personality of the individual who is responsible for the achievement.

Your Institute, like other long established scientific bodies, has an integrity beyond reproach. It does not work for money but for scientific advancement. Let its usefulness be extended to broader fields and let us resolve to make this organization, which has now been called for by the President of the United States, a useful and permanent force in our whole national life.

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REPORT OF THE SECRETARY OF THE AMERICAN INSTITUTE OF MINING ENGINEERS FOR THE YEAR 1915

I have the honor to submit herewith the report of the Secretary for the year 1915, showing the principal activities of the Institute:

Meetings.—The 110th Meeting, including the Annual Business Meeting, was held in New York City, Feb. 15 to 17, 1915. The full report of this meeting is given in Volume LI of the *Transactions*. The number of papers presented was 53, and the attendance was 357 members and guests.

The 111th Meeting was held in San Francisco, Cal., Sept. 16 to 19, 1915. Sixty-eight technical papers were presented, and the attendance was over 400 members and guests. The full report of this meeting is given in the *Bulletin* for December, 1915.*

Local Section.—A new, strong Local Section of the Institute, with headquarters in Arizona and known as the Arizona Section, was formed during 1915.

Affiliated Student Societies.—Three new Affiliated Student Societies were recognized in 1915, consisting of the Pick and Hammer Club of the University of Oklahoma, Mining and Metallurgical Club of the University of Toronto, and the Scientific Club of the Texas State School of Mines and Metallurgy.

Publications.—Full account of the publications of the Institute is given in the report of the Library Committee. It is to be especially noted that all papers are now put through the Committee on Papers and Publications and the Editorial Department of the Institute and sent to the printer not more than 30 days from the time of receipt, except in a few cases where unavoidable delays occur. Three volumes of *Transactions* were issued in 1915, namely, Volumes XLVIII, XLIX and L. The last of these was sent to members on September 30. Volume LI has been printed and will be delivered to members as fast as their dues for the year 1916 are paid.

Membership.—Full account of this is given in the report of the Committee on Membership. The membership of the Institute on Jan. 1, 1915, was 4,950. The membership on Dec. 31, 1915, was 5,214.

Honorary Member.—Professor James F. Kemp, Sc. D., LL. D., was elected an Honorary Member. A brief notice of Professor Kemp's career was given in the April *Bulletin*.

Library.—A new library agreement was made with the United Engineering Society and the other Founder Societies whereby there is a closer bond of union between the libraries than formerly, with greater efficiency in operation. The Library Board of the United Engineering Society has also carried on more effectively the work of the Library Research Bureau, under the able Chairmanship of E. Gybbon Spilsbury, whereby the benefits of the library are extended to members distant from headquarters. Full details on these points are given in the report of the Library Committee.

Additional Space for Members.—For the special benefit of members outside of New York City, additional space has been occupied by the

*See also *Trans.*, vol. lii, p. vii.

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Institute on the ninth floor and two Members' Rooms established, where are provided as complete facilities for the convenience and use of out-of-town members visiting the City as can be thought of, including reading, writing, telephoning, messenger service, and opportunities for conferences.

International Engineering Congress.—The International Engineering Congress was supported by the Institute together with four other National Engineering Societies. The proceedings of this Congress took place in San Francisco, September 20 to 25, 1915, and have been reviewed in full in the technical press.

Naval Consulting Board.—On July 19, 1915, the Secretary of the Navy addressed a letter to the President of the American Institute of Mining Engineers asking that the Institute nominate two members of the Board which subsequently became known as the Naval Consulting Board.

A meeting of the Executive Committee of the Institute was held at which it was decided that a letter-ballot be taken of the members of the Executive Committee to suggest five members of the Institute for this Board; the five names having the highest number of votes were then to be sent to the Board of Directors with the statement that the Board might vote for any of the names so suggested, or for any other members of the Institute. As a result of these two proceedings Messrs. William L. Saunders and Benjamin B. Thayer were nominated, and were subsequently appointed by the Secretary of the Navy on the Naval Consulting Board. The subsequent operations of the Board have been detailed by the daily press and also in the Institute's *Bulletin* of September, October, and December. President William L. Saunders is Second Vice-Chairman of the Board. All the officers and five additional members of the Board are members of this Institute, thus making nine members of the Institute out of the twenty-three members of the Board. The Institute has also extended to the Naval Consulting Board the privileges of its facilities, both in the way of library services and also the use of the Institute rooms for meetings. This invitation has been accepted in the same cordial spirit in which it was offered and the Naval Consulting Board used the offices of the Institute for some eleven meetings on December 22, as well as for the full meeting of the Board that same evening. The Board also held a committee meeting at Institute headquarters on December 31. On December 22 all members of the Board of Directors of the Institute who were then in the city entertained, as individuals, the Naval Consulting Board at dinner at the Engineers' Club.

Second Pan-American Scientific Congress.—The Institute is coöperating with the Department of State of the United States in carrying on the Second Pan-American Scientific Congress, full details of which have been published in the daily press and notice of which was published in the Institute's *Bulletin* for December, 1915. Mr. Hennen Jennings of the Board of Directors of the Institute is Chairman of the Committee on Mining, Metallurgy, Economic Geology and Applied Chemistry, and also a member of the Executive Committee of the Congress. The President of the Institute was in attendance at the Congress during its first week, namely, from December 27 to January 1, and the Secretary of the Institute was in attendance for the second week.

National Reserve Corps of Engineers.—The Secretaries of four National Engineering Societies invited General Leonard Wood to luncheon at the Engineers' Club on Jan. 26, 1915, at which time General Wood outlined

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a suggestion for a National Reserve Corps of Engineers. This led to action on the part of several societies of which record is given in the October *Bulletin* of the Institute. A joint committee was formed upon which this Institute is represented by Dr. Henry Sturgis Drinker, and the matter is now being deliberated by Congress.

Coöperation with Other Societies.—Besides the coöperative activities mentioned previously in this report the Institute has coöperated at meetings of the American Institute of Electrical Engineers and the Mining and Metallurgical Society of America. At the latter meeting, which was held in Washington, in December, 1915, delegates of the Institute were instructed to vote on behalf of the Institute in favor of a thorough revision of the mining laws and the establishment of a government commission to investigate and make recommendations for the proposed mining law revision. By authority of the Board of Directors the President of the Institute appointed five members of the Institute to coöperate with similar committees from other bodies in keeping before Congress the desire of the mining profession regarding changes in the mining laws, and the establishment of a permanent commission upon this subject.

The members appointed were the following: James R. Finlay, D. C. Jackling, Hennen Jennings, C. F. Kelley and E. B. Kirby.

The Institute has also been represented at meetings of several other societies and has entertained at its meetings delegates of sister societies.

BRADLEY STOUGHTON, *Secretary.*

REPORT OF THE COMMITTEE ON MEMBERSHIP FOR 1915

The total number of applications brought before the Committee during the year 1915 was 558; the total number of persons who were elected and became members of the Institute during the same period was 546.

The total membership of the Institute on Dec. 31, 1915, was 5,221, consisting of 20 Honorary Members; 4,814 Members; 210 Associate Members, and 177 Junior Members. The changes in membership during the year are shown on the accompanying schedule:

Total membership, Dec. 31, 1914		4,958	
Loss by resignation	120		
Loss by suspending	142		
Loss by death	45		
Total loss during 1915		307	4,651
Gain by election	546		
Gain by reinstatement	24		
Total gain during 1915			570
Membership, Dec. 31, 1915			5,221
Change of Status:			
Member to Honorary Member	1		
Members to Life Members	2		
Associates to Members	3		
Junior Members to Members	3		
Total	9		

J. H. JANEWAY, *Chairman.*

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REPORT OF TREASURER, 1915

Receipts

General Funds:		
Initiation fees	$4,705.00	
Arrears of dues	1,345.79	
Current dues	40,924.78	
Advanced dues	1,613.29	
Sale of binding	10,367.48	
Sale of advertising	7,109.98	
Sale of Transactions	4,085.42	
Sale of special editions	553.17	
Sale of Bulletins and pamphlets	2,028.22	
Miscellaneous Receipts:		
Returned to general funds from land fund account	146.50	
Interest on investments and deposits	383.29	
Sale of pins and fobs	278.05	
Library insurance cancelled	103.95	
Sale of library books	28.70	
Refund from Madison Affl. Student Society	11.22	
Miscellaneous	396.47	$74,081.31
Cash balance, December 31, 1914		6,316.90
		$80,398.21
Special Funds:		
Total receipts for land fund account	$10,646.50	
Returned to general funds of the Institute	146.50	
	10,500.00	
Interest on Hadfield Prize	84.70	
Interest on Thayer Prize	1.52	
Life memberships to be invested	1,050.00	$11,636.22
Cash balance, December 31, 1914		2,181.04
		$13,817.26

Payments

General Funds:		
Bulletin	$15,187.06	
Transactions	11,787.34	
Binding	9,799.41	
Special editions	47.65	
Editorial and office	25,233.63	
Treasurer	753.01	
Library	4,254.75	
Advertising	4,016.30	
Meetings	1,811.21	
Local sections	1,436.62	
Technical committees	267.14	
Back volumes	868.42	
Circulars	455.62	
International Engineering Congress	1,603.01	
Miscellaneous	1,813.65	
December 31, 1915, Cash on hand	1,063.39	
Total		$80,398.21
Special Funds:		
Final payment on land mortgage	$5,000.00	
Returned to subscribers	5,500.00	
		$10,500.00
Life membership invested		1,010.56
New York section		240.86
		$11,751.42
December 31, 1915, Balance on hand		2,065.84
		$13,817.26

GEORGE C. STONE, *Treasurer.*

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REPORT OF LIBRARY COMMITTEE FOR 1915

In accordance with the requirements of our By-Laws I beg leave to submit herewith the report of the Library Committee for the year 1915.

The activity shown by the sale of the Institute publications during the year continues satisfactory, although not as large as last year. The total sales amount to $6,666.81, distributed as follows:

Sale of Transactions	$4,585.42
Sale of Special Editions	553.17
Sale of Pamphlets and Bulletins	2,028.22

It should be borne in mind that the sale of special editions was only begun last year, and most of the members who did not have the special editions immediately bought them for their libraries. This year only a little less than half of the same number have been disposed of, and it is more than probable that from now on the number will be considerably lessened.

The additions to the Institute Library during the past year amount to a total of 913 volumes and pamphlets distributed as follows:

	Volumes	Pamphlets	Total
Gifts	104	114	218
Exchange	557	7	564
Purchase	120	2	122
Old material	3	6	9
	784	129	913

During the same period the total additions to the combined libraries of the United Engineering Society amounted to 3,214.

The attendance at the Library during the year amounted to 12,790.

During the past year the Library Board has introduced several new features which it is hoped will enable the non-resident members of our Institute to avail themselves of the Library facilities to a far greater extent than has heretofore been possible. This has been done by the establishment of a Library Service Bureau. While this Bureau was only established late last Summer the results have been extremely satisfactory, and tend to prove the necessity there is for such aid to our members.

While before the establishment of this Bureau the amount received for search work as it was then done amounted on an average to less than $250 per annum, the past year shows receipts of $2,666.89, with expenditures to obtain these results of $2,000.54. Included in the receipts, however, is the sum of $750 which the three Societies voted to the Research Bureau as a fund for the commencement of its work, so that while the large increase of receipts is satisfactory, the Bureau is still in debt to the Societies for the sum advanced to it.

During the year, the Library has at last succeeded in shelving and cataloguing all of the duplicates belonging to the American Institute of Mining Engineers. Lists of these duplicates—both periodicals and text books—have been published in the *Bulletin* and sent out to the member-

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ship at large, with notice that if any of these volumes are desired by members they can be had on application to the Library.

It is suggested by the Library Board of the United Engineering Society, and recommended by your Library Committee, that after the expiration of three months whatever volumes have not been asked for by the membership shall be disposed of to the best advantage possible either by public or private sale or, failing that, shall be donated to the libraries of Colleges or Institutions, especially those which are connected with our affiliated student branches.

Your Committee further have to report that as members of the Library Board of the United Engineering Society they have attended all the meetings during the year of that Committee.

Respectfully submitted,
E. GYBBON SPILSBURY, *Chairman.*

REPORT OF THE COMMITTEE ON PAPERS AND PUBLICATIONS FOR 1915

During the year 1915 the Committee received 161 manuscripts, which were treated as follows:

3 accepted for New York Meeting (110th)
64 accepted for San Francisco Meeting (111th)
49 accepted for New York Meeting (112th)
3 accepted for Arizona Meeting (113th)
29 rejected
4 returned to authors for revision and not yet accepted
9 in committee.

161

Every paper which was presented at the San Francisco Meeting was published in advance in the *Bulletin* with one exception.

The work of the Committee has been systematized and the Executive Committee of the Committee on Papers and Publications has given prompt attention to all manuscripts, with the result that every paper received by the Committee on Papers and Publications is in the hands of the printer within 30 days thereafter, unless delays occur owing to questions as to acceptability, in obtaining from the author permission for revision, or other like causes.

Based upon 53 papers received subsequent to February, 1915, the average time occupied by papers in the different departments is as follows:

	Days
In Committee on Papers and Publications	14
In Editorial Department	18
Composition, correction, and furnishing of proof	9
Transmission to author and return, correcting proof, and final printing	22
Total	63

While the time in the Editorial Department given above may seem somewhat out of proportion, the Committee reminds the readers of this report that the *Bulletin* is published only once a month; thus some papers are unavoidably held in the Editorial Department for several days no

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matter how rapidly editorial attention is given to them; other papers require extensive editorial revision, involving correspondence with authors, etc., and these incidents make the average time longer than it otherwise would be. The Committee also reminds readers of the report that the Editorial Department and the Committee on Papers and Publications are occasionally under a great pressure of papers, especially just previous to a meeting of the Institute, with the result that papers have to wait their turn before receiving attention.

BRADLEY STOUGHTON, *Chairman.*

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PAPERS

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Notes on Flotation*

BY J. M. CALLOW,† SALT LAKE CITY, UTAH

(New York Meeting, February, 1916)

Historical Sketch

The selective action of oils for lustrous minerals was first disclosed by Haynes in 1860. In 1885, Miss Carrie Everson elaborated this idea and also disclosed the fact that acid increased the so-called selective action. Her patent called for oils, either animal, vegetal or mineral, also for an acid or salt. The process was tried out on a practical scale in Baker City and Leadville, in 1889, and failed: first, because, as has since been shown, of the unsuitability of the ore to flotation; and second, because the invention was too far in advance of the times. Then followed the Elmore brothers, first with their bulk oil processes and later with their vacuum method. The basic principles of oil flotation were undoubtedly covered by the above inventors, the work done since their time being merely a building up on ground work laid by them. Different kinds of oil, different quantities of oil, and all the varying degrees of agitation, were all exemplified and practiced by them in one phase or another, and the developments that have since been made are but elaborations of the fundamental principles laid down by Haynes, Everson, and the Elmore brothers.

In 1902, the Potter-Delprat process was developed in Australia. In this process no oil was used, the mineral being raised by the generation of gas, brought about by the introduction of acid into the pulp. The mineral particles appeared on the surface of the separatory vessel, in the form of a scum or froth, buoyed up by minute gas bubbles attached to them. This no doubt first gave the suggestion of gaseous flotation.

In 1902, also, Froment, an Italian, was granted a patent in which he combined violent agitation with oil and gaseous flotation, the gas being generated within the pulp, in much the same way as in the Potter-Delprat process.

In the same year, Cattermole came out with a unique method. First, the pulp was emulsified with a small quantity of oil, by violent

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*Originally presented at the annual meeting of the Utah Section, Salt Lake City, Oct. 4, 1915.

†Consulting Engineer.

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agitation, and then submitted to a slow stirring action in a second machine. This granulated. or coagulated, the minerals that had been oiled into nodules which were afterward separated from the pulp by gravity The defect of this process was that only part of the mineral granulated, the rest of it appearing on the surface of the pulp as a scum or froth, and so being lost in the tailings. This defect of the Cattermole process suggested the fundamental idea of the process afterward described by Sulman, Picard, and Ballot in their patents, in which, instead of granulating part of the mineral, all of it was floated. This patent forms the basis of all the Minerals Separation operations. It was first exploited in Australia and in a short time replaced all other flotation processes in that country.

In 1904, MacQuisten brought out his ingenious tube process, which gave excellent results on the sandy portions of the feed, but was inoperative when slime was present. This was a strictly surface-tension method, and its inability to handle slime was a serious limitation.

In 1912, Hyde introduced a modification of the Minerals Separation process into the mill of the Butte & Superior Co., Butte, Mont. This differed from the regular practice in that roughing and cleaning of the concentrates were done.

Description of Callow Pneumatic Flotation Process

Early in 1909, I did a great deal of work with the MacQuisten flotation process, and was instrumental in the tube-plant installation at the Morning mill at Mullan, Idaho. This work was followed by a large amount of experimenting on the different kinds of existing flotation processes, the outcome of which was the development of the pneumatic process.

The first application of pneumatic flotation for the treatment of ore was made at the mill of the National Copper Co., at Mullan, Idaho, which was designed and built by me. Construction was started on Aug. 14, 1913, and the plant went into operation about Apr. 10, 1914. The flow sheet is shown in Fig. 1. It was a metallurgical success in every way from the start.

Since that date, the process has been adopted by nearly all the other mills in the Coeur d'Alene treating lead and lead-zinc ores, notably the Gold Hunter, Morning, Hercules, Bunker Hill & Sullivan, Caledonia, Last Chance, Hecla, Standard, etc., a total of about 50 cells in all, treating from 1,500 to 2,000 tons of slime and fine sand feed per day. The same process too has since been adopted by the Inspiration Copper Co., the Arizona Copper Co., the Anaconda Copper Mining Co., the Magma and other copper companies, and by the Silver King, Daly-Judge, Duquesne, and El Rayo mining companies, on lead, zinc and other ores, making a total of some 680 cells operating, or in the course of installation, having a combined capacity of about 25,000 to 28,000 tons per day.

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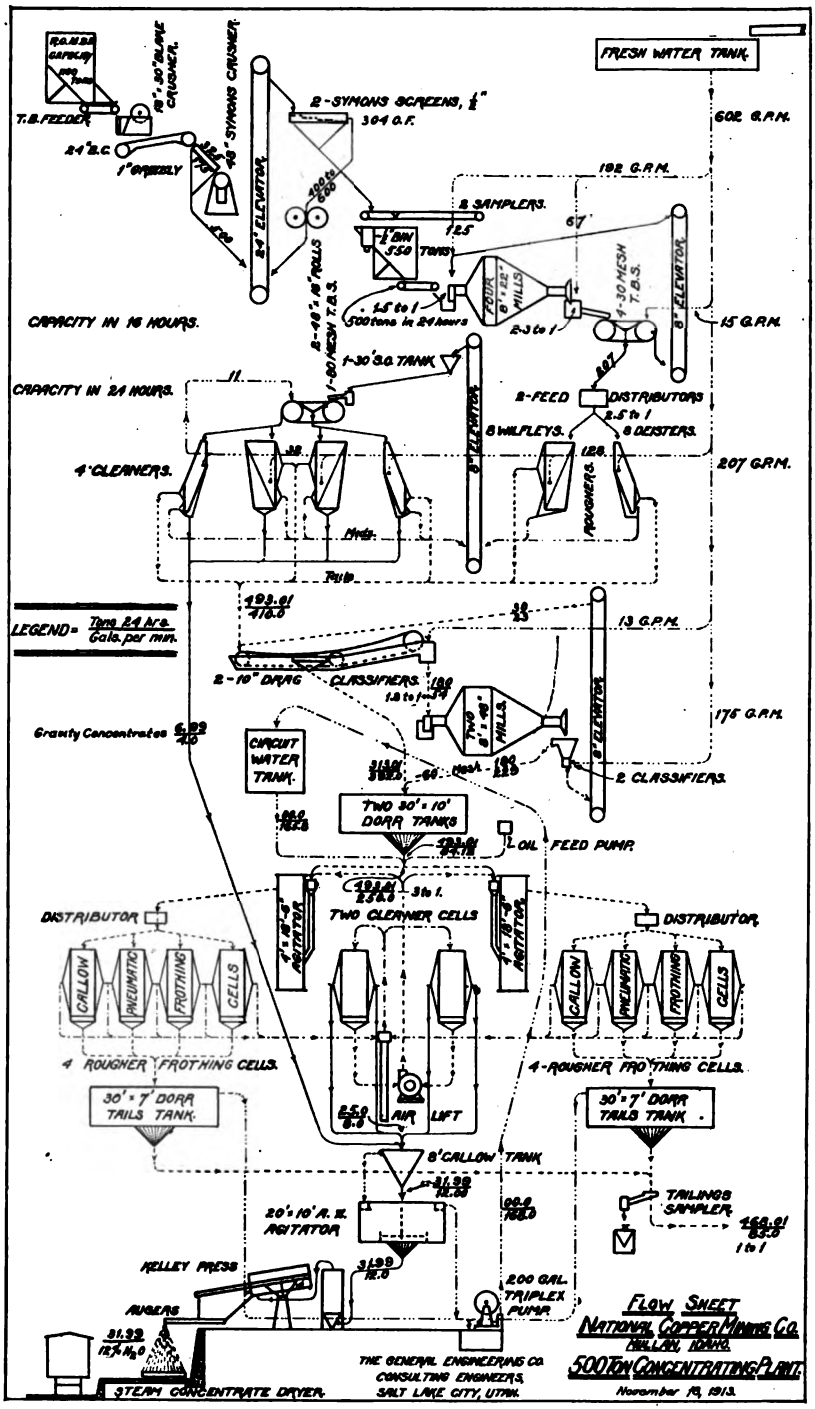

FIG. 1.

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Fig. 2 illustrates the various elements composing the Callow process in general.

Mixing

In the mixer *A*, operated by compressed air, the oil, air, and water are mixed and emulsified, the same type of apparatus being in common use in cyanide works. In cases where the oil, or frothing agent, can be fed into the crushing machine or tube mill, this mixer, or Pachuca tank, can be dispensed with, the tube mill discharging direct to the separatory cell.

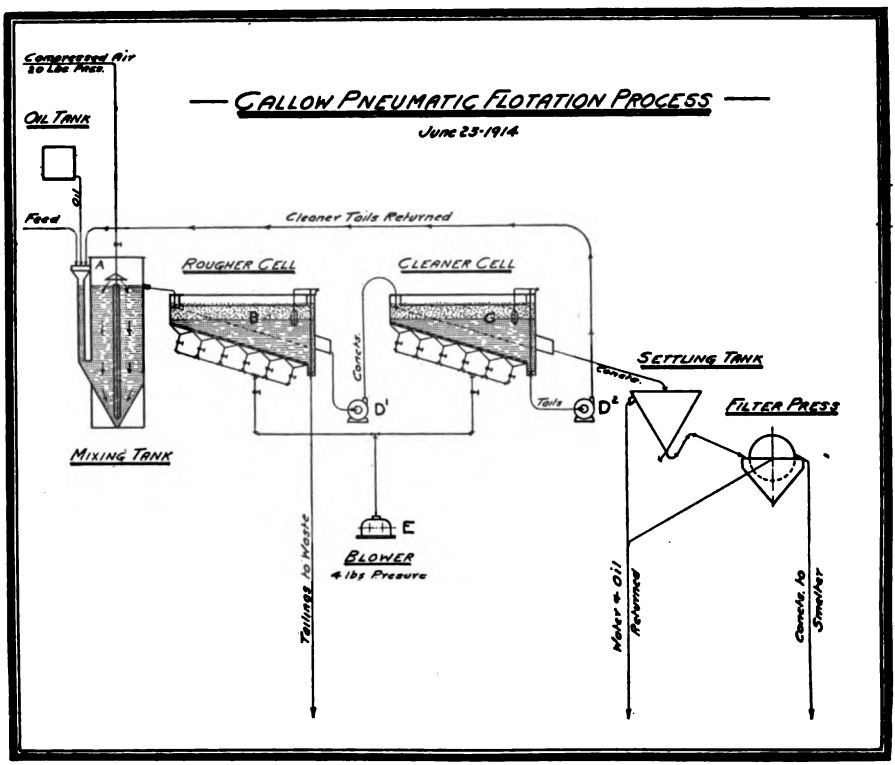

FIG. 2.

It has been conclusively proved that agitation *per se* is not necessary to successful flotation by the pneumatic method. In one of the plants a Pachuca mixer for each four roughing cells was installed. The pulp from a Dorr thickener was delivered to the Pachuca by a belt-and-bucket elevator, the oil being fed into the boot of the elevator. It was found that the mixing obtained in the elevator alone gave as good results as when the Pachucas were used; so the use of Pachuca tanks in this plant was abandoned.

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The Separatory Cell

The initial or roughing separatory cell *B* consists of a tank about 9 ft. long overall, and 24 in. wide, with a bottom inclined at from 3 to 4 in. to the foot; it is 20 in. deep at the shallow end and 45 in. deep at the other end. It may be built of either steel or wood, but wood construction is preferable.

Fig. 3 shows the cell in detail. The bottom of the tank consists of a porous medium made of four thicknesses of loosely woven canvas twill, properly supported by a backing of perforated metal to prevent its bulging when under air pressure. Through this porous medium compressed air is forced by the blower *E* (Fig. 2). Porous brick, or any other ceramic material that will give the necessary fine subdivision to the air, may also be used. Some of these have been tried out, but for practical and mechanical reasons the loosely woven canvas twill seems to serve all purposes better than anything else, and has been adopted as the standard porous-bottom construction.

The space underneath this porous medium or bottom is subdivided into eight compartments, each connected by an individual pipe and valve with the main air pipe. By this means the air pressure to each compartment can be regulated (by throttling the valve) to correspond to the varying hydraulic head within the tank, and so as to discharge a uniform amount of air throughout the length of the bottom and maintain a uniform aeration of the contents. A pressure of from 4 to 5 lb. is generally used, each square foot of porous medium requiring from 8 to 10 cu. ft. of free air per minute.

Each longitudinal edge of the tank is provided with a lip and an overflow gutter for the reception of the froth to be discharged. The lower end of the tank is furnished with a spigot discharge fitted with a plug valve, operated by a float, to maintain a uniform water level within the tank and thus, in turn, maintain a uniform and constant discharge of froth under all the varying conditions of feed supply incident to practical milling operations. The water level may, of course, be varied, but is usually maintained at about 10 to 12 in. below the level of the overflow lips.

The tailing is discharged through the spigot and the frothy concentrate is conveyed by means of the side gutters to the pump D^1 and thence to the cleaner separatory cell *C*. This cleaner cell is a machine of the same construction as the rougher. In operation, however, it is usually run with a lower air pressure than that used on the rougher. The tailing from the cleaner is returned by pump D^2 to the original feed, a closed circuit thus being maintained on this portion of the feed. The concentrate from the cleaner is the shipping or finished concentrate. Pump D^1 can well be eliminated by setting the cleaner at a lower elevation and

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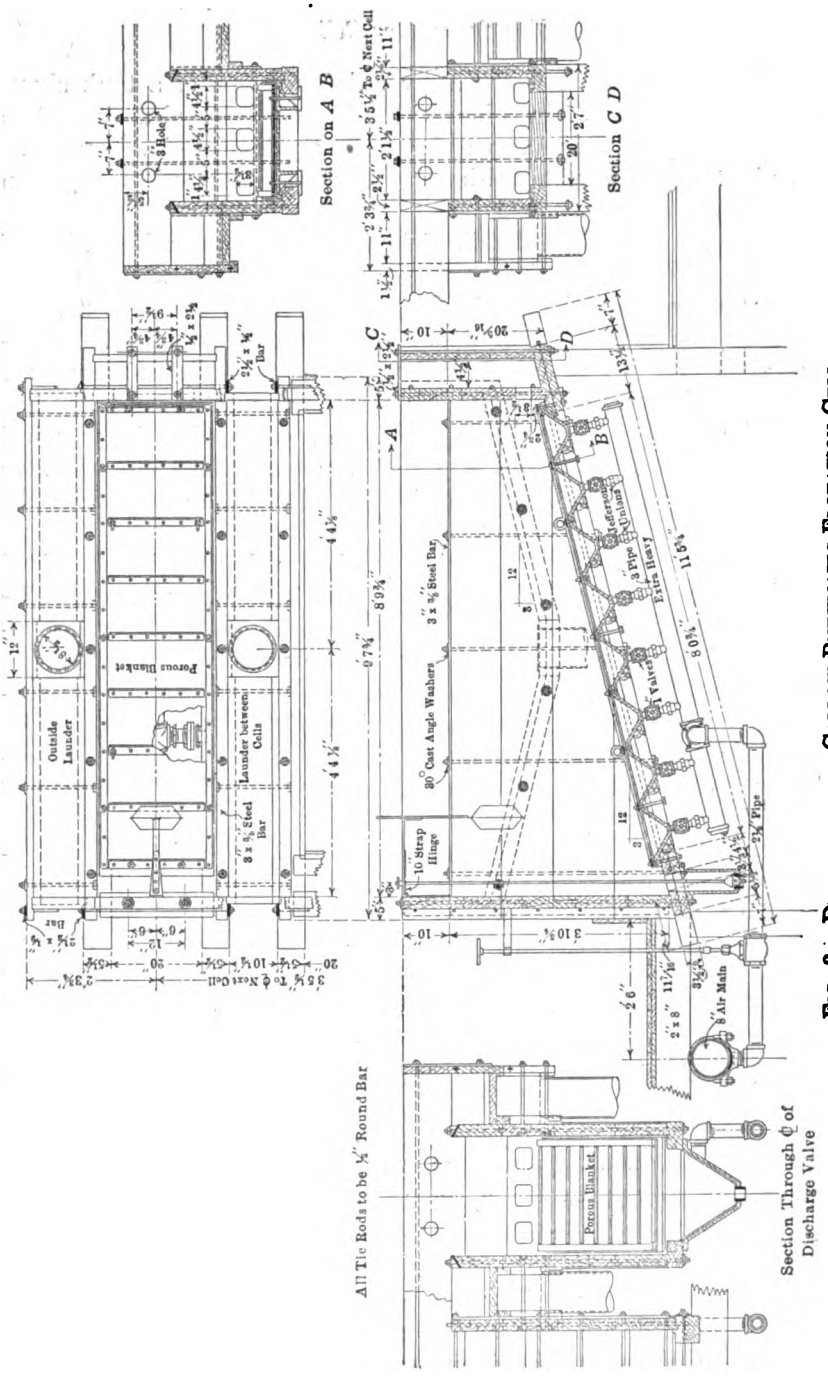

FIG. 3.—DETAILS OF THE CALLOW PNEUMATIC FLOTATION CELL.

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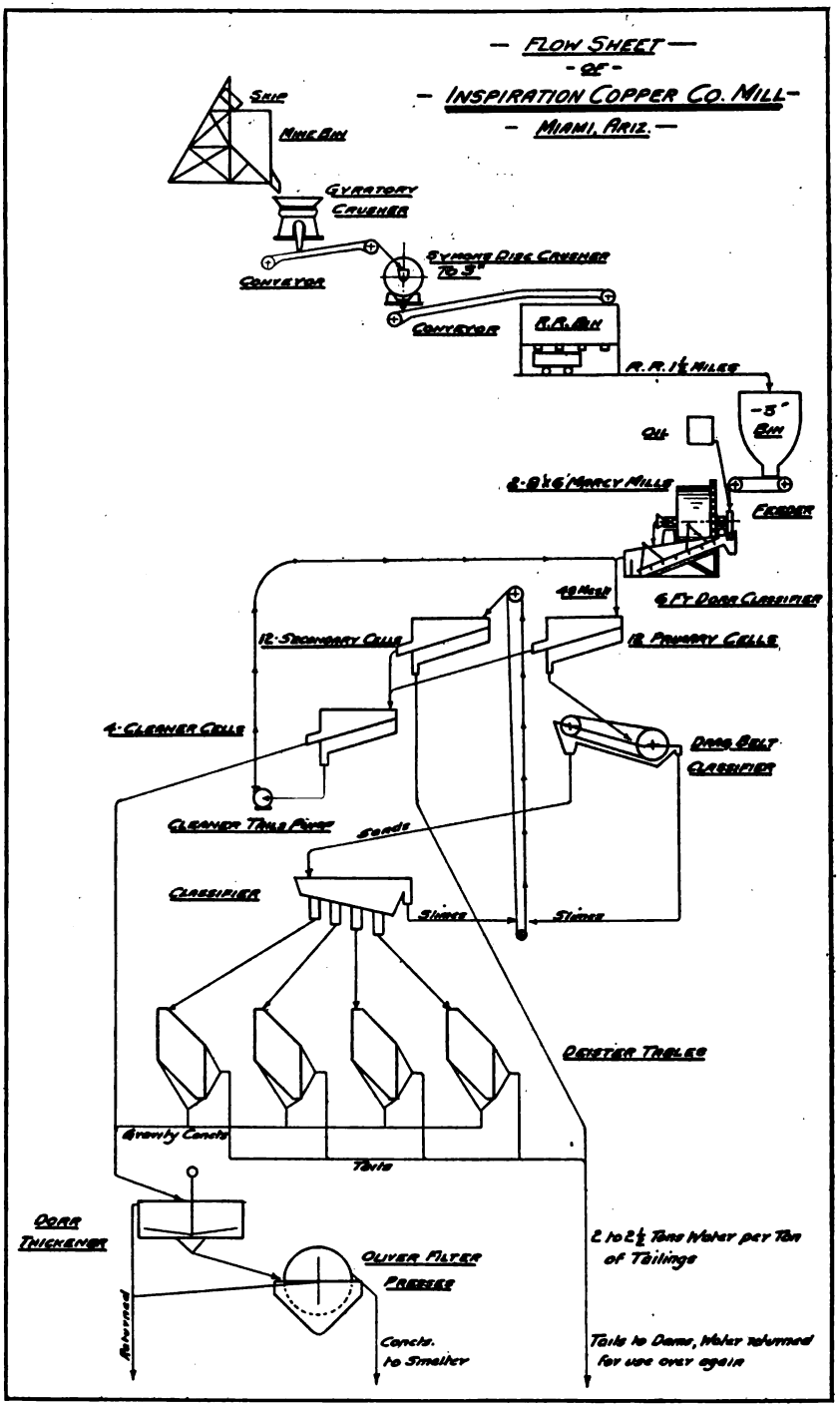

FIG. 4.

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conveying the rougher froth to it by gravity. Usually one cleaner serves four roughers.

Cells May Be Used in Parallel or Series

The machines may be run either in parallel or in series without any sacrifice in the capacity for a given number of cells. Recent experience goes to show that on some ores the series treatment gives a slightly cleaner tailing; on others it does not. It is not necessary to extend this arrangement of cells beyond two cells in series. In a heavily mineralized ore this arrangement is decidedly advantageous, and in such a case the rougher concentrate might be of high enough grade to dispense with the re-cleaning operation. The froth from the second cell in the series might be returned to the original feed in the same way that the tailing is returned from the cleaner when practicing a roughing and cleaning operation. A number of such combinations are possible.

At the mill of the Inspiration Copper Co. (Fig. 4), the original feed goes to 12 primary roughers, the tailing from which is classified into sand and slime, the sand going to tables and the slime to 12 secondary roughers. The concentrates from both the primary and secondary roughers go to four cleaner cells, and the tailing from the cleaner cells is pumped back into the circuit.

Froth Formation

The froth is generated as the result of injecting the finely divided air into the bottom of the already emulsified pulp; it continues to form and to overflow so long as it is furnished with pulp of the proper consistency, adequately mixed with the right quantity and kind of oil or frothing agent. Measured from the water level within the tank, the froth produced may be from 14 to 16 in. thick and will be more or less voluminous, coarse or fine grained, dry or watery, according to the character of the ore and the kind and quantity of oil introduced. The condition of the froth may be varied therefore, by changes in the kind and quantity of oil used, and the quantity of air injected.

In some ores, rich in sulphides, and where a comparatively low-grade concentrate will suffice, the cleaning cells may not be necessary, but on low-grade ores having a high ratio of concentration and when a concentrate of extreme cleanness and of maximum grade is required, a cleaner is desirable.

Pulp Density

The pulp to be treated may be of varying density, from 2½ of water to 1 of ore, up to 5 or 6 to 1. For a mixture of sand and slime, the former ratio is preferable, but for a pure slime mixture (minus 200 mesh) the

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larger proportion of water is allowable. The particular density is not a matter of as much importance as the supplying of pulp of uniform density, since each variation in the density of the pulp requires a readjustment of the oil supply, the quantity of oil increasing in proportion to the increased volume of the pulp, independent of its solid contents.

Capacities

A normal capacity for each standard roughing cell is 50 tons per 24 hr. This, of course, will vary according to the nature of the ore. In one plant which practices gravitation previous to flotation, only the fine sand and slime being treated, the rate is 50 tons per rougher cell. The Inspiration Copper Co. practices flotation as the prime process, each 800-ton section employing 24 roughing cells and four cleaners. The cells in this case are run in series, the primary cells treating the original feed and the secondary cells treating only the slime from the primary tailing after the sand has been removed. This gives an average of 33.3 tons per roughing cell. The Arizona Copper Co.'s plant (Fig. 5) will treat the slime and re-crushed sand from previous gravity treatment; out of an original tonnage of 4,000, about 3,600 tons will be treated by flotation. This will be handled on 63 roughers run in parallel, and 18 cleaners, or an average of approximately 57 tons per roughing cell, or 45 tons per cell for roughing and cleaning.

Some tests have shown little difference in recovery, whether running 45 tons to the cell or 65; but that the recoveries commence to decline as soon as the tonnage exceeds 75 tons. In the Coeur d'Alenes, on zinc-lead ores, 35 tons per cell is an average capacity. The flow sheet of the Daly-Judge mill (Fig. 6) shows the arrangement for treating a zinc-lead ore in the form of a fine sand and slime feed.

Oils Used

The oils used may be broadly divided into "frothers" and "collectors." The pine oils are good frothers and coal tar and its various subdivisions are good collectors. On some ores, crude pine tar will in itself combine the properties both of frothing and collecting; on others, this may have to be enriched by the addition of some one of its more volatile constituents, such as refined pine oil, turpentines, or wood creosote.

Generally speaking, the coal-tar products are poor frothers, and to get a sufficient volume of froth to insure a high recovery it is often necessary to add refined or crude pine oil, creosote, etc. At the Inspiration mill, for instance, the mixture is 80 per cent. crude coal tar, 20 per cent. coal-tar creosote; at another plant on similar ore, 45 per cent. El Pasc coal tar, 40 per cent. coal-tar creosote, 10 per cent. cresol, and 5 per cent.

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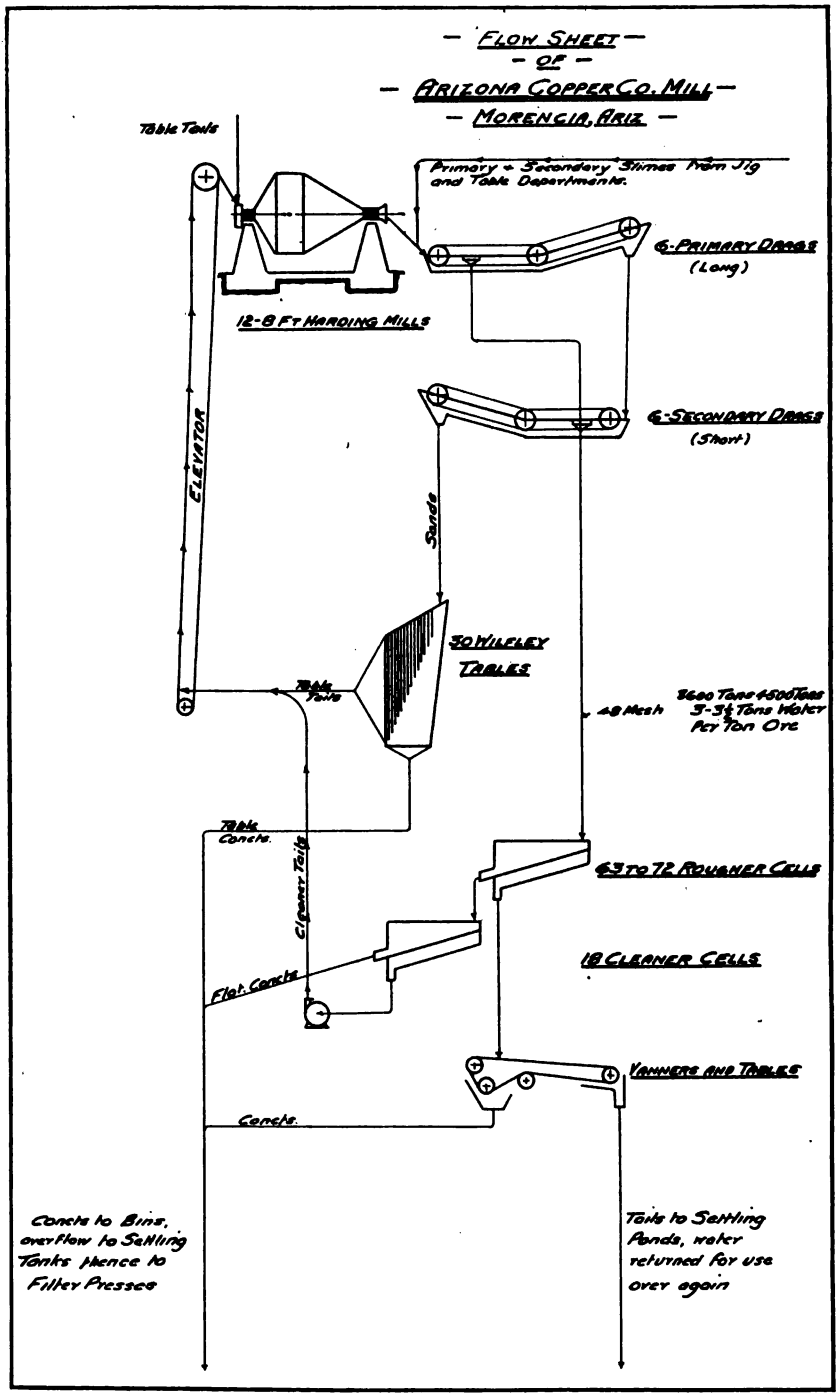

FIG. 5.

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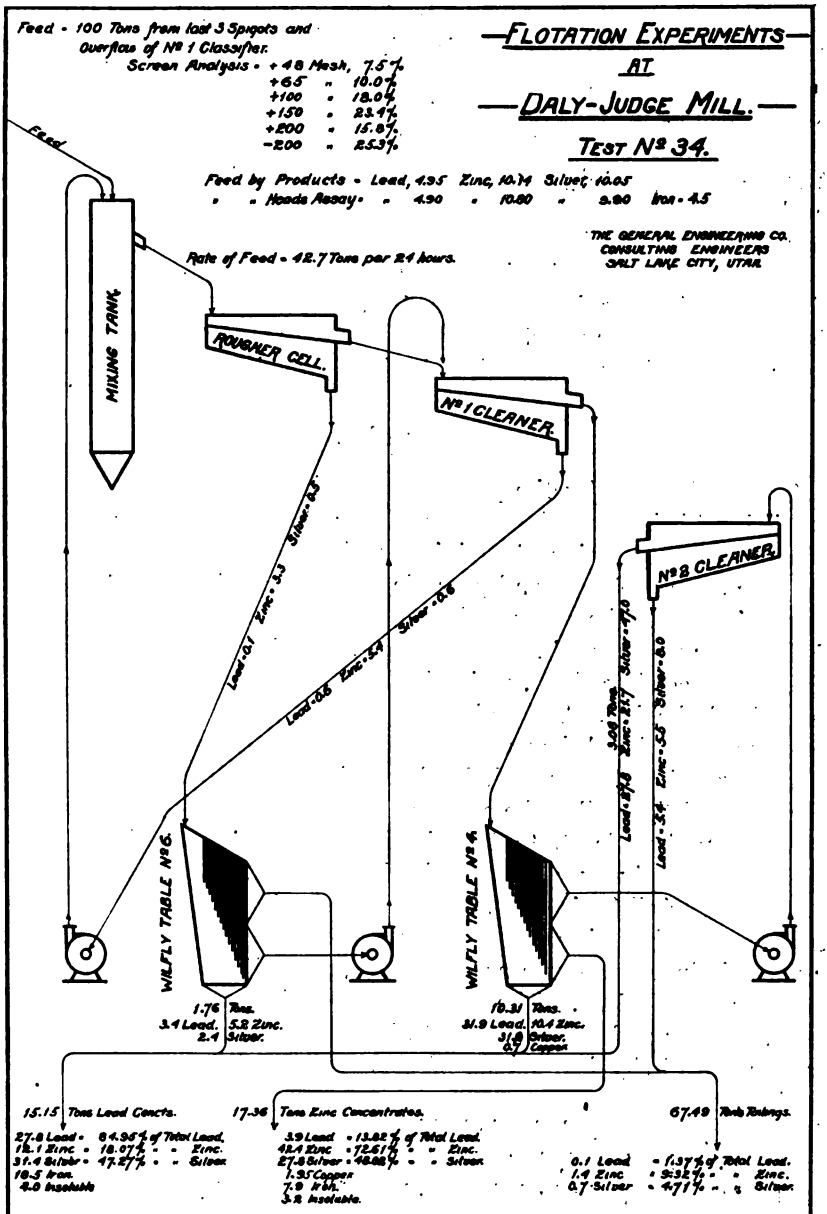

FIG. 6.

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pine oil. At Daly-Judge a mixture of 40 per cent. crude coal tar, 40 per cent. creosote, and 20 per cent. pine oil was used. In the Coeur d'Alene on zinc ore straight wood creosote was used; on the ore of the National Copper Mining Co., plain turpentine will work, but pine oil is better.

At Inspiration, from 1½ to 2 lb. of the mixture per ton of ore was used; at Daly-Judge, 1 to 1½ lb.; and at the National 0.3 lb. of oil is sufficient. In the experimental work at another plant the oil consumption was approximately 1 lb. of mixture per ton; but since the entire plant has been in operation, and the circuit water reclaimed and used over again, the oil consumption has dropped to 0.35 lb. At present the proper kind or kinds of oil and the requisite quantity can only be determined by experiment; no scientific way has thus far been found.

Character of Froth

The froth made by the pneumatic process has the distinctive characteristic of being unstable or ephemeral; that is, it quickly dies when removed from the action of the injected air. The bubbles composing the froth are generated under a hydraulic pressure varying from 15 to 40 in.; upon rising above the water to the froth level, the bubbles burst by reason of the lower pressure of the atmosphere. On bursting, they release the mineral attached to them, which is caught up by the bubbles following immediately behind. The stability of the bubbles depends to some extent upon the oil used and the nature of the gangue in the pulp treated. Pine oil makes a brittle froth which dies immediately on arriving at the surface. Creosote and light oil make a more elastic envelope which at times will expand into bubbles 3 to 4 in. in diameter before bursting. Pine-oil bubbles are rarely over ¼ or ½ in. in diameter. Castor oil, olive oil, candle-makers' oil (oleic acid), palm oil, sperm oil, and other oils of a lubricating nature, have in general been replaced by oils more or less soluble or miscible in water—such as turpentine, pine oil, and all the coal- and wood-tar distillations. The extremely volatile oils such as naphtha, gasoline, ether and alcohol, seem to be of little use except as means for making the pitchy ingredients of coal and wood tars more soluble or miscible.

A large, coarse, and elastic bubble seems necessary to the recovery of coarse-grained mineral, but for the very fine and colloidal mineral, a small and comparatively brittle bubble is necessary.

Power

The National Copper Mining Co., using approximately 950 cu. ft. of air at 4-lb. pressure, and treating 500 tons per day on eight roughers and two cleaners, required 35 hp.; this shows a requirement of 3.5 hp. per cell, equivalent to 12.53 tons per horsepower or 1.25 kw.-hr. per ton.

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Another company using approximately 9,600 cu. ft. of air at 5-lb. pressure and treating 2,400 tons per day on 48 roughers and 12 cleaners, requires 210 hp., equivalent to 3.5 hp. per cell, 11.45 tons per horsepower, or 1.56 kw.-hr. per ton.

The experimental plant of the Inspiration Copper Co., using approximately 950 cu. ft. of air at 5-lb. pressure and treating 200 tons per day with four roughers and one half-size cleaner, required 24 hp.; deducting 4 hp. for two 2-in. centrifugal pumps the figures become 4 hp. per cell, 10 tons per horsepower, or 1.79 kw.-hr. per ton.

A maximum figure would be 2½ kw.-hr. per ton of feed, using 5- to 5½-lb. air pressure, generated by Roots or Connersville positive blowers.

Costs

Oil.—The oil mixtures generally in use cost from 1.25c. per pound up to 3c. per pound., depending on the percentage of cresol and other high-priced oils used; on most ores, 1½c. per pound is a safe average figure, and a consumption of 1 to 1½ lb. or from 1.25c. to 4.5c. per ton of feed, averaging 2½c. would be a safe estimate.

Labor.—This will vary, of course, with the size of the plant. At one plant consisting of 60 cells, two men per shift operate the entire plant, equivalent to a cost of 1¼c. per ton. One man per shift on a 250-ton plant means a cost of 5.4c. per ton maintenance. Assuming a life of three months per blanket, a capacity of 50 tons per cell, and an allowance for repairs to blowers, motors, pumps, etc., we have ½c. per ton as a liberal allowance.

Power.—With power at 1c. per kilowatt-hour, and a consumption of 2½ kw.-hr. per ton, the cost would figure 2.5c. per ton of feed.

Summarized, the estimated cost on a 2,000-ton flotation plant, or larger, would be approximately as follows:

	Per Ton, Cents
Labor	1.25
Oil	2.50
Maintenance	0.50
Power	2.50
Total	6.75

On a plant of 250 tons the extra labor costs per ton would bring it up to approximately 10c. per ton of flotation feed.

Actual figures from a large plant treating over 2,000 tons by flotation gave 6.1c. per ton; the flotation feed in this case represents 60 per cent. of the crude-ore tonnage, making the cost 3.5c. per ton of crude ore treated.

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Theories

So far no satisfactory explanation of flotation phenomena has been advanced. At my instigation and under my direction, a large amount of research work has been done in an endeavor to formulate some logical explanation of the phenomena, and perhaps to find some scientific way of conducting flotation experiments in place of the empirical methods now in vogue. Although the latter object has not yet been attained, still these experiments have resulted in the formulation of a theory that appears to be well grounded and that may prove of interest and value to others engaged in this art.

Much work has been done at the Mellon Institute at Pittsburgh under the direction of Dr. Raymond C. Bacon, and lately by James A. Block at the station of the U.S. Bureau of Mines in Salt Lake City. The result of all of this work is summed up in the following statement:

In considering the connection between flotation phenomena and the physical properties of the minerals concerned, there are two parallelisms:

First.—It has been noticed for some time that the minerals which floated were not easily wetted by water, while those which were easily wetted did not tend to come up with the froth. This is the basis of about the only theory so far widely circulated; it was well stated by Hoover in his book, *Concentrating Ores by Flotation.*

Second.—There is a parallelism between certain electrostatic characteristics and the flotation properties of ores.

In the first-mentioned theory, surface tensions and contact angles should be considered. Certain minerals, such as galena, will float on the surface of still water, while gangue particles, since they possess a greater adhesive attraction for the water than the water's cohesive attraction for itself, will be drawn through the surface film into the interior, and sink because of their greater specific gravity. These properties of floatable minerals and gangues are increased by the presence of oil and acid. Oil sticks to galena with greater tenacity than it does to silica, and an oil surface is still less easily wetted than a galena surface. The acid in the water causes a still greater difference in the various surface tensions. This, it seems, is without question the explanation of the action in the MacQuisten process, in which the ore particles are lifted to the surface, where they can be removed by skimming off the surface layer of the liquid.

With reference to the second parallelism, it has been noticed that extremely small amounts of certain colloidal impurities, such as saponin or tannin, were detrimental to flotation; while others, such as Congo red and methylene blue, did not interfere, and were, if anything, beneficial. In classifying these, the injurious ones generally came under the head of what physical chemists call electronegative colloids, while electropositive colloids were not harmful. Suspended particles will generally migrate

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when placed in an electric field, and this classification comes naturally from the direction of their migration. This migration is called electrophoresis, or electrical endosmose, and is the result of the formation of contact layers around the particles by the liquid containing them, very similar to the formation of surface films when liquids come in contact with air. These contact films almost invariably have a difference of potential between their inner and outer surfaces. An air-water contact film has, for instance, a difference of 0.055 volts, and other contact films have similar charges. This causes the particles to act like charged solids, and to be attracted by electric charges of opposite sign.

The charges on solids and non-miscible liquids can be conveniently studied on the stage of a microscope.

This work naturally led to the study of the charges exhibited by various ores and minerals, and in that work an interesting parallelism was observed; namely, that floatable minerals seemed to have positive charges, and non-floatable gangues negative charges. Some gangues were found with positive charges, but they were characteristically hard to handle, having a tendency to come up with the froth. These charges sometimes vary with the acidity or alkalinity of the liquid, and this variation is not inconsistent with the effects of acidity or alkalinity on the flotation of ores.

It has been noticed that these electrostatic properties depend on the condition of the surface of the particle and not upon the composition of the mass. For instance, lead oxide which is ordinarily negative, or neutral, when covered with a sulphide coating, takes upon itself a positive charge.

Recent investigations on the coagulation and deflocculation of slimes, on the coagulation and dispersion of colloids, and along similar lines, show that these contact film charges, although small, have an important bearing on the dispersion or coherence of particles suspended in liquid mediums. In fine suspensions and in colloidal solutions, these charges may often be neutralized by the introduction of oppositely charged ions, precipitation generally taking place whenever these charges fall below certain limits. Oppositely charged contact films have a general tendency to coalesce, while similarly charged films, if their charges are great enough to overcome natural cohesiveness, do not seem to coalesce, but to repel each other, and if the weight of the particles is small enough in relation to their size and surface, permanent dispersion will take place, the particles distributing themselves through a liquid in much the same manner that gas will fill a container.

In view of the above observations, it seems possible that flotation is the result of difference in polarity in the charges on the various ore particles, and on the bubbles. Since oil contact films and air contact films have both been proved to have negative charges, the positively charged miner-

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als will adhere to either. The bubble mantles in a flotation machine are undoubtedly composed of oil, or of oil in emulsion, since pure water alone will not froth. The same forces, then, that cause oppositely charged colloids to agglomerate and precipitate, cause the minerals to adhere to the oil-covered bubbles; and the same forces that keep the particles of an oil emulsion dispersed, keep the gangue particles repelled from the bubbles.

Expressed briefly, the theory is as follows:

That oil flotation is an electrostatic process. It is a scientific fact that when a solid particle is suspended in water, the water will form around the particle a contact film which generally possesses an electric charge, the amount and polarity depending upon the nature of the surface of the particle and the electrolyte in which it is suspended. The presence of these charges can be demonstrated by the fact that the particles possessing them will migrate when placed in an electric field. It has been demonstrated that floatable particles have charges of one polarity (positive), and that non-floatable particles have charges of the opposite polarity (negative); that the froth is charged negatively and so attracts the positively charged or floatable minerals, and repels the negatively charged or non-floatable ones. It is this, it is believed, that causes the floatable minerals, galena, sphalerite, etc., to adhere to the froth, and the gangue minerals, silica, etc., to remain in the liquid where they can be discharged as tailing.

Discussion

James A. Block, Salt Lake City, Utah.—As I have done some of the experimental work upon which this theory has been based, a brief outline of some of my experiments might not be out of place. During the last few months I have tested about 50 samples of ores, etc., the flotation characteristics of which I knew. Some of these were samples of froth and tailings from plants in actual operation, but most of them were samples taken from laboratory tests going on at the plant of the General Engineering Co., and at the station of the U. S. Bureau of Mines, Salt Lake City. In all of these experiments, no real exception to the theory was found. All of the tailing samples produced in actual flotation showed a negative polarity; all of the ores with the so-called "bad gangues," except one or two, showed some positive polarity, and these exceptions could be explained by the extreme dryness and scaliness of the gangue minerals; and all of the froth products showed positive polarity when the samples were fresh. It was often true that after standing some time, these froth samples would reverse, especially when they contained considerable oil, but this reversal can be explained if we assume that the oil forms a coating around the particles upon standing.

The question might naturally be asked, "can these electric charges

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be put to any practical use?" A machine[1] used to purify clays employs very similar charges on the constituent particles of clay. This machine (see Fig. 7) consists of a revolving cylinder, made of conducting material, such as iron, about half immersed in a clay pulp in a trough. At a distance of about ½ in. from the immersed side of the cylinder is a coarse wire screen. A direct current of about 50 volts was applied to the cylinder and the screen, the cylinder being the anode and the screen the cathode. The cylinder attracted the negatively charged kaolin, while the screen attracted the positively charged pyrite, ferric hydroxide, and aluminum hydroxide. It was found that a product running only about 17 to 20 per cent. moisture could be scraped off the upper side of the cylinder, which revolved slowly, while the impurities fell through

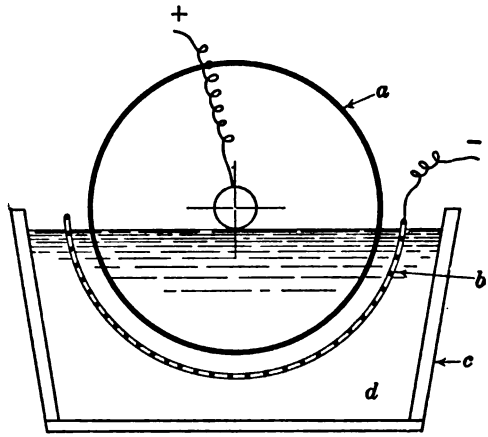

FIG. 7.—APPARATUS USED IN PURIFYING CLAYS.

a Revolving cylinder
b Wire screen
c Wood trough
d Clay pulp

the screen to the bottom of the trough. The objection to the process was the high cost of the power needed, which averaged, as I remember it, over 30c. per ton of product, but the process is undoubtedly of interest in that it shows that these electrostatic charges are dependable enough to be put to practical uses. It is also of interest in that the polarity of the charges noticed on various minerals checks the results of the experimental work upon which this theory is based.

OLIVER C. RALSTON, Salt Lake City, Utah.—Perhaps the most interesting part of Mr. Callow's paper is the reference to a theory of flotation founded on the electric charges of the ore particles, oil droplets, air bubbles, etc., involved.

[1] *Transactions of the English Ceramic Society*, vol. xii, pp. 36 to 64 (1912-13).

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For some time I have had a growing conviction that possibly the particles dealt with in the flotation of sulphide minerals were acting according to certain laws laid down in colloid chemistry. One of the characteristic things about either suspension or emulsion colloids is that their individual particles are charged with one or the other sign of static electricity. It is known that the amount and signs of these charges can be controlled by the amount and character of electrolytes put into the water in which the colloids are dispersed, and in turn the properties of the colloidal suspension are greatly changed with this change in the electric charges on the particles. The question has been whether the comparatively coarse suspensions of ore can be compared with suspension colloids, or whether they were too coarse to allow the electric charges on the suspended particles of mineral to produce any measurable effect.

This question has been answered fairly definitely in my laboratory during the past year, for it was found that the electric charges on the particles of what we call "slimes" and "fines" are of considerable importance in the control of the rate of settling of such slimes in water. With this much definitely settled, and with a great amount of literature available in the realm of colloid chemistry dealing with the electric charges on particles of such things as quartz, galena, and sphalerite, and on oil droplets and air bubbles, it was easy to find a combination of circumstances which might explain flotation from this new viewpoint. At about this time George W. Riter made the prediction that the depths of electrostatics would have to be thoroughly sounded before flotation phenomena could be entirely understood. He seemed to have the idea at that time that the charges on the particles were generated by friction in the flotation machine, in much the same way that a piece of sealing wax becomes charged when rubbed with flannel; but we know from colloid chemistry that these charges of electricity arise whenever two substances of unlike dielectric constants come into contact.

Further work along the line of controlling the charges of the ore particles, thereby controlling the rate of settling of the particles in water, has convinced me that the charges are of considerable importance and the results of this work will be reported soon. Mr. Callow told me last spring that he had much the same conception of the matter as I, and had the Mellon Institute in Pittsburgh working on it for some time. Mr. Callow sent a man to the Salt Lake laboratory of the U. S. Bureau of Mines to try out some methods of measurement which I had proposed and which had not been used in Pittsburgh; some of the results are presented in Mr. Callow's paper. His work adds certainty to the conjecture that the electric charges on particles of minerals suspended in water are of importance in flotation phenomena. Mr. Callow has outlined one way of explaining just how these electrical effects come into play in flotation and I have outlined a second in an article in the *Mining*

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and Scientific Press. The knowledge that when flotation conditions are good the froth has a positive charge and the rejected gangue a negative charge, that the oil droplets and the air bubbles are negatively charged, and the sulphide particles positively charged, is highly interesting. We have here electrical effects which are parallel to flotation phenomena, and the conclusion that they are connected with each other seems justifiable. Just exactly how they are connected is not yet clear and may prove difficult of solution.

It is to be hoped that further work will be done along this line in order that a theory of flotation may be found, which will afford a more rational control of flotation testing, and an extension of the process to the treatment of any kind of minerals.

R. H. RICHARDS, Boston, Mass.—I think there is one note which would be interesting to have recorded in connection with this paper, and that is another form of flotation machine. The Minerals Separation machine has a vortex action, with a vertical shaft and a propeller. This new frother has been developed for the Copper Queen Co. at Bisbee, Ariz. It has a horizontal shaft and a paddle wheel for a frother. It has eight cells, I think, all connected underneath in one continuous tank, and the only separation of the cells is in the upper chambers where the froth is. When this paddle wheel goes around it throws out the froth and the sand goes to the bottom and drifts over from the first to the second cell, where it is paddled again, and comes out with its froth; the sand goes to the third cell where it is again beaten. In that way the sand is continually drifting toward the last cell, while each new froth remains in its own cell. You can see in the working of this machine just the same effect that you see in Callow's machine, *i.e.*, the earlier cell has the darker or richer froth, and the latter has the lighter or poorer froth until you get to the farthest one—which has very little froth, and is very light.

JOSEPH W. RICHARDS, So. Bethlehem, Pa.—Speaking from the metallurgical standpoint, it is quite desirable, in many cases, that the oil be removed from the concentrate in order to facilitate the metallurgical work, and I think attention should be directed to the desirability of removing from zinc sulphide concentrates the oil which interferes with their subsequent roasting. The value of the oil is not large; in Mr. Callow's plant, for instance, it would be $20 a day, and it may be possible that a process might be developed which, within the limits of the value of the oil, would remove it, and thus facilitate the metallurgical treatment.

LEONARD WALDO, New York, N. Y.—The question came up about a year ago as to why Mexican crude was not an admirable oil for flotation. I have never heard any satisfactory answer to the question.

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The Mexican crude has the peculiar property of comparative freedom from dangers of fire, a very great reluctance to part with its water, and a specific gravity almost identically the same as water.

The great difference between the Mexican crude, the Lower California crude, and the Texas crude, leads one to be unwilling to apply any tentative results obtained with either the California or the Texan crude to the Mexican oils, which are essentially different in the relation of paraffin and asphaltum in their base. I would suggest to some of those interested that it would be well worth trying to see how far that is applicable. The future price of Mexican oil will undoubtedly return to the neighborhood of 2¼ c. per gallon, which would make it perhaps one-eighth of the cost of the oil specified here, and the fact that the coal-tar oils, which are closely similar in their general behavior to the Mexican crudes, have been used, makes one think it is a promising source of investigation.

GEORGE D. VAN ARSDALE, New York, N. Y. (written discussion).—To many of us, the most interesting part of Mr. Callow's paper is that pertaining to the theory of flotation.

About a year and a half ago we started a line of research to prove or disprove a theory of flotation, which was purely electrostatic, and quite similar to the theory presented by Mr. Callow. Since then we have put a considerable amount of work and study on the subject which has led to a modification of our original theory, our present ideas being about as follows:

The immediate cause of the main phenomenon of flotation is the resultant of the several surface tensions involved, together with the wetness or non-wetness of the particle, but electrostatics is probably very important as a secondary cause, which may and probably does act in at least two different ways: (1) as one of the causes of wetness or non-wetness: and (2) as one of the possible causes of oil attraction to mineral particles.

Now, if we define flotation as being a process of concentration, utilizing the varying behavior of the surfaces of particles to the surfaces of liquids and of gases, we can make a classification of the various methods into four classes:

I. Utilizing varying behavior of surfaces with respect to the surface of one liquid.

II. Utilizing varying behavior of surfaces with the surfaces of one liquid and of gas bubbles.

III. Utilizing varying behavior of surfaces with the surfaces of two immiscible liquids.

IV. Utilizing varying behavior of surfaces with the surfaces of two or more immiscible liquids and of gas bubbles.

It seemed probable that the basic principles underlying all these

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classes were sufficiently similar so that the investigation of the simplest class would throw considerable light on the others. We therefore started with Class I. It seems obvious enough that the immediate cause of a substance specifically heavier than water floating on the surface of the water is the surface tension involved. If this is so there should be a definite relation between the surface tension of water and the maximum weight that can be supported on a given surface by reason of surface tension, and this relation should be capable of experimental proof. It has been said that this maximum weight is equal to the surface tension of water, but this easily can be shown to be incorrect. The relation, however, as suggested by Mr. Langton, can be derived from the surface tension formulæ for the differences in pressure between two sides of a curved liquid surface. For water and for a spherical and cylindrical surface these two formulæ are:

1. (Sphere) $P = \frac{2T}{R}$

2. (Cylinder) $P = \frac{T}{R}$

where P = pressure
T = surface tension
R = radii

By making assumptions as to the contact area, by the use of formula 2, we can calculate the maximum weight, for example, of a cylinder of copper wire that can be supported on the surface of water. The results of the calculations give No. 13, 16 and 18 B. & S. gage for 180°, 120° and 60° contact area, respectively. Now, experimentally, perfectly clean No. 17 gage copper will just float on pure water; No. 16 will float when slightly greased, and No. 15 when greased. Since, therefore, our experiment gives results so close to theory, we can consider as proven that one of the main causes of this class of flotation is surface tension. There are of course other conditions and among these is shape of particle. We can show mathematically from our two formulæ that the maximum weight of a cylindrical particle will be twice that of a spherical particle, both having equal radii and degree of wetness. Another condition is, of course, size of particle, since we can easily show that many substances otherwise incapable of flotation may float if powdered to a sufficient degree of fineness, although in some cases this flotation is imperfect and temporary. Finally, a most important condition of this class of flotation is the inherent property of being non-wet, possessed by some substances, and of the addition to other substances of films, some of which promote flotation, others preventing it. It is easy to show experimentally that such films may be either a solid, a liquid or a gas, or an electric film, that is, an electrostatic charge, or a combination of these. If then we summarize these conditions we can make the following statement:

Any substance specifically heavier than water may be made to float

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by reason of surface tension provided its size be small enough, and provided the particle be non-wet, that is, has a sufficient surface film, which may be either solid, liquid or gas, electrostatic charge, or a combination of these. This is strikingly similar to a fundamental principle of colloids. Now in our Class II, we have the additional factor of gas bubbles, and the additional phenomenon to explain of why the gas bubble attaches itself to the mineral particle and not to the gangue particle. In order to explain this, let us picture a system of two particles, one wet and the other non-wet, and a gas bubble, all three in contact. Let us magnify these largely and consider what we have. You see at once that we have a surface of liquid in contact with a surface of gas and two particles at the surface, one wet and the other non-wet, or, in other words, precisely the conditions of Class I, and we know from our study of this just what will actually take place, which is that the wet particle will sink, while the non-wet will float or remain in contact with the bubble surface. The fact that the whole system is small, or movable, does not alter the relative surface tensions involved. The kind of gas, or manner of application, apparently makes little difference, it being necessary only to bring a bubble of gas into contact with a non-wet particle.

Now in Class III we have the phenomena that some particles in contact with oil and water have the property of entering and remaining in the oil, others of entering and remaining in the water, and others again of remaining in the boundary surface separating the two. It seems to me that as pointed out by Ralston we have an adequate explanation of these phenomena in the work of Reinders, who explains them on the basis of the inequalities of interfacial tensions.

Going on to Class IV (our modern flotation methods) we have the varying behavior of particles with respect to the surfaces of two or more immiscible liquids and of gas bubbles. Let us consider the states or conditions in which the oils and gas bubbles may be present in a flotation machine:

Oil may be present: (1) as emulsion; (2) as globule; (3) as film on surface or bubble surface.

Bubble may be present: (1) as clean air bubble; (2) as bubble with oil film; (3) as bubble in contact with oil globule.

Now let us consider what will happen in case a non-wet mineral particle comes into contact with any of these:

If the particle comes into contact with an oil globule, the conditions of Class III are met.

If the particle comes into contact with a clean air bubble, the conditions of Class II are met.

If the particle comes into contact with a bubble with oil film, the conditions of Class II are met, modified by the reduced surface ten-

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sion of the bubble surface caused by the oil film. Any combination of these conditions can of course take place.

The remaining case—when a particle of mineral meets an oil in the emulsified condition—is interesting, since we have apparently proved that at least under certain conditions there is in this case no attraction apparent between the oil and the mineral; in other words, the force or condition producing emulsification is greater than any attraction between the oil and the mineral. Also, in this respect we have apparently proved that at least one of the functions of sulphuric acid is to prevent such emulsification, which is a well-known property of acid in other organic work, for example, grease rendering. We have also found a number of substances much more efficient in this respect than sulphuric acid. From the converse of this proposition—that if emulsification of an oil is undesirable in that when in this condition its attraction for sulphides is at least greatly reduced—we would expect to find certain substances capable of producing emulsions and thereby being very deleterious in practical work. It is, of course, well known that glue tannin, saponin, etc., are capable of entirely destroying flotation action, and it seems to me very significant that all of these substances belong to the class of emulsifying agents, some of them being extraordinarily efficient, and that probably their known deleterious action is entirely due to this effect.

The main point which I have tried to bring out is that in my opinion we will have not two more or less opposed theories, one "surface tension" and the other "electrostatics" but we will finally have a theory made up of both, with electrostatics and the study of colloids to furnish our explanation of many of the basic principles, but with surface tension as our immediate course.

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Grinding Brass Ashes in the Conical Ball Mill

BY ARTHUR F. TAGGART* AND R. W. YOUNG,* NEW HAVEN, CONN.

(New York Meeting, February, 1916)

FOREWORD

THE tests herein described are part of an extended series of experiments, performed by the authors together with J. F. McClelland and L. W. Bahney, on the reclamation of metallics from foundry and manufacturing-plant scrap. The most obstinate problem encountered was the proper cleaning of the metal.

INTRODUCTION

In the manufacture of brass and brass products the weight of the finished product leaving the plant is much less than the total weight of copper and zinc entering. A small part of this discrepancy is due to the unavoidable oxidation of the metals in melting, but by far the largest part of the loss is in the form of metallic particles of copper, zinc, and brass. These losses occur principally at three points in the process: (a) in the casting-shop ashes; (b) in the slags from furnaces melting scrap brass; (c) in material spilled on the floor throughout the manufacturing process. The amount of these products in some plants is nearly 100 tons per 24 hours.

Casting-shop ashes contain the metal spilled from the crucibles during melting and pouring, and the metal included in the slags formed in the melting process. The ashes also contain pieces of broken crucibles with adhering metal; 20 to 30 per cent. of unburned coal is also present. The metal in casting-shop ashes contains clean pieces of zinc or copper ingots weighing 1 to 2 lb., jagged pieces of metal of all shapes ranging in size from several inches down to the finest dust, and shot and botryoidal shapes ranging in size from 2 or 3 in. down to 0.01 mm. or less in diameter.

Slags from furnaces melting scrap brass contain metal in the form of small shot.

Floor sweepings occur in all shapes: spirals, thin shavings, pins and pin-like pieces, wire, rods, and buffing-wheel dust are the commonest.

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In general the problem of reclaiming this metal has been to free the metal from adhering slag and separate it from the associated slag, coal and cinders. On its face this problem appears simple. That it is not simple, however, is attested by the concentrations of fine metallic particles in the sewers and stream beds of the New England brass-manufacturing districts. The usual plant installed has consisted of a crude hand-picking installation for recovering the coarse metal; a restricted-discharge grinder, such as the Huntington or Chilian mill, Krupp ball mill, Hill cinder crusher, etc., for severing the metal from the slag; a concentrating table, usually a Wilfley, for treating the fine discharge from the grinder; and a settling tank for the slimes flowing away from the concentrating table. The coarse metallics are collected intermittently by stopping the plant and cleaning out the grinding machine. These plants are claimed to throw away tailings assaying but 0.5 per cent. copper. Such statements are based on erroneous samples, for the tailings will, when the plant is properly sampled, be found to contain up to 5 per cent. copper, and most of the coal is wasted.

In connection with the design of a plant for the treatment of brass ashes the writers were confronted with the problem of finding a grinder that would answer the following requirements: (a) take a feed containing particles of metal 2 or 3 in. in diameter, with adhering slag, and at the same time pieces of slag containing metal shot in all sizes from the above down to the finest; (b) discharge continuously both the metallic and non-metallic constituents of the feed, and thus eliminate the intermittent factor attendant upon the operation of the grinders previously mentioned; (c) discharge a product of such character that all the material larger than 2.5 mm. is clean metal, while all the slag and cinders are ground to pass the same screen; (d) have a capacity of about 1 ton per hour; (e) have a reasonable repair and power charge. The following are the results of two tests run on a 4½-ft. by 16-in. conical ball mill with these requirements in mind.

Description of the Test

The mill used in these tests was a special one built by the Hardinge Conical Mill Co. for testing purposes and installed in the Hammond Mining and Metallurgical Laboratory, Sheffield Scientific School, Yale University. As will be seen from Fig. 1, the mill is mounted with the discharge end *a* on a fixed foundation *b*. The feed end *c* is carried on the wooden pier *d* which can be slid parallel to the long axis of the mill and fastened in any desired position by holding-down bolts in the floor slots *e*. The mill can be broken along the rims *f* and the cylinder *g* removed, thus giving a mill with no cylindrical section, or one, two, or three cylinders can be inserted making the cylindrical portion of the mill 16 in., 32 in., and 48 in. long respectively, as desired. The mill is lined with chrome-

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steel lifting bars. For the tests described herein, the following are the mill adjustments:

One 16-in. cylinder used, making the mill the standard 4½ ft. by 16 in.; tilt, 2.5 in. toward the discharge end; speed, 28 to 29 r.p.m.

Ball Charge:

Size of Balls, Inches	Number of Balls	Weight, Pounds
5	100	1,927
4	100	951
3	205	815
1¾	1,002	720
Total		4,413

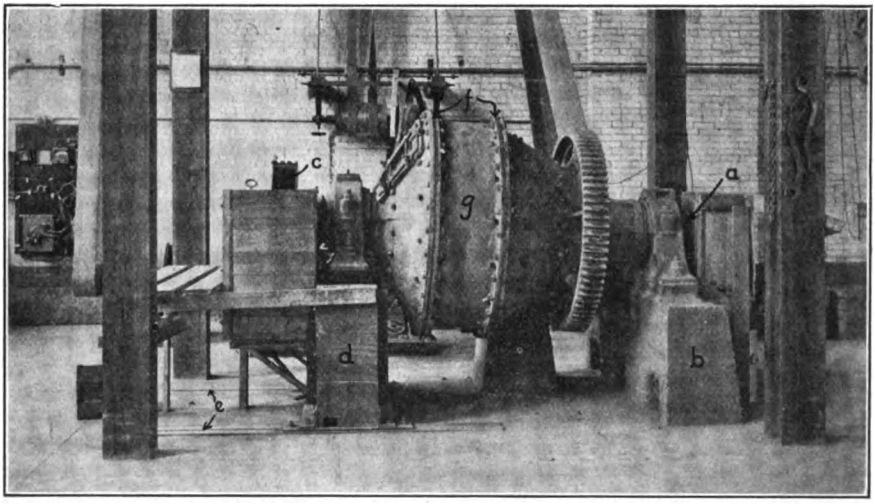

FIG. 1.

Details of Tests.—Two runs were made, which are referred to as No. 1 and No. 2 in the following notes.

Run No. 1.—Feed, sluice concentrate, containing 30.01 per cent. coal. (For sizing test see Table I.)

Feed rate, 2.24 tons solid per hour.
Moisture in feed, 42.9 per cent.
Power consumption, 20 hp.

Run No. 2.—Feed, first- and second-gate discharges from jig, containing 1.23 per cent. coal. (For sizing test see Table III.)

Feed rate, 0.91 tons solid per hour.
Moisture in feed, 39.7 per cent.
Power consumption, 21 hp.

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Products.—Sizing-sorting-assay tests of feed and product of the two runs are given in Tables I, II, III, and IV.

Summary of Results

1. A concentrate can be made by simply screening the discharge from the mill. The grade of this concentrate will depend on the character of the feed to the mill and the aperture of the screen used. Since brass containing more than 10 per cent. "dirt" (slag and coal) is unsuitable for remelting, the concentrate obtained by screening the product from Run No. 1 is not sufficiently clean. The product from Run No. 2 can be screened on a 1.651-mm. screen and will give a concentrate containing 90.9 per cent. clean metal. If cleaner metal is desired, it can be obtained by using a screen of larger aperture. (See Tables II and IV.)

2. The material passing the 1.651-mm. screen in Run No. 2 (see Table IV) containing 13.9 per cent. metal, is too rich to be thrown away, as is done in some plants, or to be sold to a junk dealer at a couple of dollars a load, as is done in some others.

3. In Run No. 2 about 25 per cent. of the total feed to the mill, containing 70 per cent. of the total metallic in the feed, can be saved clean enough for remelting without the use of concentrating machines. (See Table IV.)

TABLE I.—*Sizing-Sorting Test on Feed. Run No.* 1

On Screen					Tons per 100 Tons				Per Cent. Clean Metallics in Product on Screen
Aperture, millimeters	Weight, Grams	Tons per 100 Tons	Cumulative Per Cent.	Assay Cu, Per Cent.	Free Metallic	Slag	Contained Metallics	Coal	
18.85	670.00	1.14	1.14	20.5	0.285	0.335	0.090	0.520	25.0
13.33	4,350.00	7.87	9.01	20.8	1.516	3.680	1.104	2.674	20.1
9.423	10,260.00	18.60	27.61	11.4	3.255	9.710	0.168	5.635	20.1
6.680	10,510.00	19.05	46.66	17.7	5.229	8.592	0.176	5.229	22.0
4.699	7,295.39	13.22	59.88	23.9	4.100	5.950	0.952	3.170	24.0
3.327	7,681.80	13.92	73.80	17.2	3.638	6.840	0.197	3.442	24.4
2.362	5,081.21	9.21	83.01	17.0	1.576	5.060	0.926	2.574	23.6
1.651	2,655.15	4.81	87.82	13.2	0.529	2.763	0.485	1.518	22.9
1.168	1,517.14	2.75	90.57	10.4	0.254	1.135	0.202	1.361	22.5
0.833	1,223.11	2.21	92.78	6.2	0.103	0.749	0.117	1.358	22.1
0.589	1,107.47	2.00	94.78	4.8	0.107	1.125	0.047	0.768	21.7
0.417	780.74	1.41	96.19	8.6	0.147	0.710	0.047	0.553	21.6
0.295	641.08	1.16	97.35	8.3	0.081	0.621	0.072	0.458	21.4
0.208	433.23	0.78	98.13	8.8	0.064	0.421	0.045	0.295	21.3
0.147	326.12	0.59	98.72	4.7	0.027	0.367	0.017	0.196	21.2
0.104	289.23	0.52	99.24	8.9	0.061	0.295	0.013	0.164	21.1
0.074	125.09	0.23	99.47	9.8	0.026	0.144	0.010	0.060	21.1
Through 0.074	292.95	0.53	100.00	10.6	0.085	0.354	0.005	0.091	21.1
Loss	80.29								
Totals	55,320.00	100.00	.....	.....	21.083	48.851	4.673	30.066	

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TABLE II.—*Sizing-Sorting Test on Product. Run No. 1*

On Screen		Tons per 100 Tons	Cumulative Per Cent.	Assay Cu, Per Cent.	Tons per 100 Tons				Per Cent. Clean Metallics in Product on Screen
Aperture, millimeters	Weight, Grams				Free Metallic	Slag	Contained Metallics	Coal	
13.33	15.12	0.240	0.240	17.5	0.067	0.023	0.000	0.150	27.9
9.423	62.80	0.995	1.235	44.6	0.692	0.060	0.018	0.243	61.4
6.680	211.78	3.360	4.595	44.9	2.306	0.449	0.104	0.605	66.7
4.699	326.68	5.180	9.755	48.3	3.942	0.539	0.061	0.699	71.6
3.327	401.79	6.368	16.143	42.6	4.038	1.036	0.298	1.294	68.4
2.362	436.05	6.912	23.055	41.2	2.829	2.232	0.400	1.851	60.2
1.651	425.68	6.752	29.807	25.7	1.437	3.672	1.335	1.643	51.4
1.168	441.61	7.003	36.810	12.5	0.662	2.956	0.733	3.384	43.4
0.833	498.17	7.900	44.710	10.5	0.313	5.083	1.008	2.504	36.4
0.589	550.18	8.725	53.435	8.1	0.488	4.923	0.637	3.314	31.4
0.417	563.52	8.939	62.374	8.0	0.635	4.780	0.503	3.524	27.9
0.295	481.63	7.641	70.015	6.6	0.342	4.396	0.471	2.903	25.4
0.208	385.10	6.105	76.120	7.3	0.154	3.772	0.559	2.180	23.5
0.147	362.65	5.754	81.874	6.8	0.091	3.796	0.535	1.867	22.0
0.104	328.29	5.207	87.081	6.5	0.078	3.571	0.467	1.558	20.7
0.074	98.28	1.560	88.641	6.7	0.003	1.374	0.159	0.178	20.4
Through 0.074	715.80	11.359	100.000	9.3	0.011	8.329	1.672	2.519	18.1
Loss	104.88								
Totals	6,410.00	100.000	.....	.....	18.093	51.491	8.960	30.416	

TABLE III.—*Sizing-Sorting Test on Feed. Run No. 2*

On Screen		Tons per 100 Tons	Cumulative Per Cent.	Assay Cu, Per Cent.	Tons per 100 Tons				Per Cent. Clean Metallics in Product on Screen
Aperture, millimeters	Weight, Grams				Free Metallic	Slag	Contained Metallics	Coal	
26.67	323.00	7.810	7.810	55.9	6.285	1.525	0.698	.....	80.5
18.85	363.0	8.775	16.585	29.2	1.524	7.251	2.588	.....	47.1
13.33	1,065.00	25.757	42.342	25.8	6.045	19.591	5.425	0.121	32.7
9.423	1,088.00	26.307	68.649	31.6	8.340	17.846	4.970	0.121	32.6
6.680	720.00	17.420	86.069	36.1	7.257	9.800	2.808	0.363	34.2
4.699	380.00	9.189	95.258	42.9	5.200	3.747	1.115	0.242	34.6
3.327	91.33	2.210	97.468	43.5	1.219	0.896	0.320	0.095	36.8
2.362	28.16	0.681	98.149	43.9	0.380	0.267	0.098	0.034	36.2
1.651	19.91	0.482	98.631	37.6	0.221	0.246	0.069	0.015	37.0
1.168	9.90	0.239	98.870	32.7	0.098	0.125	0.026	0.016	37.0
0.833	6.27	0.152	99.022	25.7	0.033	0.102	0.030	0.017	37.0
0.589	6.00	0.145	99.167	21.4	0.025	0.096	0.024	0.024	37.0
0.417	4.58	0.111	99.278	17.1	0.018	0.068	0.013	0.025	36.9
0.295	4.86	0.118	99.396	15.3	0.015	0.077	0.013	0.026	36.9
0.208	4.63	0.112	99.508	10.7	0.001	0.084	0.018	0.027	36.8
0.147	4.73	0.114	99.622	10.5	0.002	0.079	0.018	0.033	36.8
0.104	5.00	0.121	99.743	7.4	0.001	0.084	0.014	0.036	36.8
0.074	2.73	0.066	99.809	9.1	0.001	0.051	0.008	0.014	36.7
Through 0.074	7.91	0.191	100.000	13.1	0.010	0.161	0.029	0.020	36.7
Totals	4,135.01	100.000	.....	.....	36.675	62.096	18.324	1.229	

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TABLE IV.—*Sizing-Sorting Test on Product. Run No. 2*

On Screen		Tons per 100 Tons	Cumulative Per Cent.	Assay Cu, Per Cent.	Tons per 100 Tons				Per Cent. Clean Metallics in Product on Screen
Aperture, millimeters	Weight, Grams				Free Metallic	Slag	Contained Metallics	Coal	
9.423	79.00	1.316	1.316	62.5	1.316	0.000	0.000	0.000	100.0
6.680	233.00	3.881	5.197	61.8	3.827	0.054	0.014	0.000	99.0
4.699	374.00	6.229	11.426	61.3	6.075	0.138	0.040	0.016	98.3
3.327	308.50	5.138	16.564	56.4	4.382	0.703	0.248	0.053	94.3
2.362	328.50	5.475	22.039	57.7	4.850	0.592	0.205	0.033	92.9
1.651	248.00	4.131	26.170	53.1	3.310	0.686	0.197	0.135	90.9
1.168	183.90	3.063	29.233	45.8	2.080	0.875	0.168	0.108	88.4
0.833	175.90	2.930	32.163	36.3	1.399	1.387	0.302	0.144	84.7
0.589	243.90	4.066	36.229	20.2	0.960	2.721	0.352	0.385	77.8
0.417	308.00	5.132	41.361	12.5	0.553	3.837	0.477	0.742	69.5
0.295	428.10	7.135	48.496	9.3	0.215	6.024	0.841	0.896	59.7
0.208	432.80	7.211	55.707	8.6	0.213	5.880	0.784	1.118	52.4
0.147	460.80	7.680	63.387	10.7	0.233	6.338	1.085	1.109	46.1
0.104	507.00	8.446	71.833	9.9	0.183	7.222	1.163	1.041	41.4
0.074	299.30	4.987	76.820	11.5	0.161	4.262	0.756	0.564	38.5
Through 0.074	1,393.20	23.180	100.000	11.5	4.265	17.067	.....	1.848	34.0
Loss	206.10								
Totals	6,210.00	100.000	.....	.....	34.022	57.786	6.632	8.192	

TABLE V.—*Elutriation Test on Minus 0.074-mm. Material, Product of Run No. 2*

Current Mm. per Sec.	Weight, Grams	Tons Per 100 Tons Original Product	Per Cent.	Cumulative Per Cent.	Assay Cu, Per Cent.	Tons Per 100 Tons of Original Product		
						Metallic	Slag	Coal
9.5	0.45	0.241	1.09	1.09	51.1	0.197	0.044	0.000
5.1	2.80	1.499	6.80	7.89	23.5	0.563	0.936	0.000
4.2	1.67	0.896	4.06	11.95	9.0	0.130	0.766	0.000
3.2	4.25	2.273	10.32	22.27	12.1	0.440	1.833	0.000
2.3	3.65	1.955	8.87	31.14	10.0	0.313	1.617	0.025
1.5	3.87	2.071	9.39	40.53	9.6	0.317	1.547	0.207
0.9	4.99	2.672	12.13	52.66	9.8	0.419	1.932	0.321
0.6*	9.09	4.869	22.09	74.75	9.9	0.771	3.528	0.570
A	10.12	5.435	24.67	99.42	10.4	0.905	3.929	0.601
B	0.24	0.128	0.58	100.00	0.0	0.000	0.095	0.033
Totals	41.13	22.039	100.00	.....	.....	4.055	16.227	1.757

* The discharge from the elutriation apparatus with 0.6 mm. per second current was led into the bottom of a 2-liter wide-mouth bottle. A siphon dipping about 1 in. into the neck of this bottle carried the fine slime to other bottles rigged up in the same way. After standing several hours the supernatant liquid in these bottles was decanted and evaporated to dryness giving the product marked *B*; the material which settled in all but the first bottle is marked *A*. The material which settled in the first bottle is that opposite the 0.6 mm. per second current.

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4. Comparison of the sizing tests on feed and product of Runs Nos. 1 and 2 shows that finer grinding was done in Run No. 2. This is due partly to the lower rate of feed and partly to the large percentage of coal in the feed in Run No. 1. It will be noted (Table II) that more than 50 per cent. of the non-metallic material remaining on the 2.362-mm. screen is coal. A comparison of results shows that the coal does not grind as freely as the slag.

5. Analysis of the data given shows that the metallics have been reduced in size (6.08 mm. to 3.62 mm. average size in Run No. 1, and 11.83 mm. to 3.01 mm. average size in Run No. 2). The reduction is not, however, so great as indicated, by reason of the fact that the largest pieces of metal were held back in the mill, the duration of the runs not being great enough for them to work through. The writers believe that the sharp increase in the percentage of minus 0.074-mm. metallics in Run No. 2 is due, not to sliming, but to the freeing of the included metallic from the slag, as was not done in Run No. 1. This belief was borne out by an elutriation test (Table V) which showed no metallic in the finest sizes, and was confirmed by microscopic examination of the minus 0.074-mm. material, which revealed the majority of the free metallics to be rounded, and not jagged as would be expected had they been abraded from the larger pieces.

6. The tables show the reduction in size of the slag in the two runs and the necessity of this reduction in order to free the metal. Microscopic examination of a slag particle, 0.1 mm. in diameter revealed the presence of metal shot not more than 0.01 mm. in diameter. In order to free these shot for concentration the greatest possible reduction of the slag is essential.

7. The reduction in size of coal is not so great in Run No. 1 as in Run No. 2. In both cases, however, the bulk of the coal will pass a 1-mm. screen, and is, therefore, too fine for burning economically without special appliances. Its value to the plant is much less than if it were removed from the ashes before grinding.

Conclusions

1. The 4½-ft. by 16-in. conical ball mill, operated as described in Run No. 2, if fed a dirty brass concentrate containing slag and a little coal, at the rate of 1 ton per hour, will grind the slag and coal so that a great part of it will pass a 1.651-mm. screen while the brass in the feed will be only slightly reduced in size and can be separated from the dirt by screening. The screening operation can be performed by attaching a trommel of the proper aperture to the discharge end of the mill.

2. If the feed to the mill contains a considerable percentage of coal, the product saved on the screen will contain more than 10 per cent.

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"dirt." The writers believe that this will be true even if the feed rate is considerably below 1 ton per hour.

3. As compared with the other types of machine at present being used for grinding brass ashes, the conical ball mill should show a decreased repair cost, since screens are eliminated, and it has a further advantage in that it discharges continuously not only the ground slag and coal but also the cleaned metal.

4. The metal is not slimed in the mill, but that which is not caught clean on the screen is of such size that it can be easily saved by proper table treatment.

5. While the basis for the conclusion does not appear from these tests, the writers have found in other work along similar lines that with the mill horizontal the metal is not cleaned up and the large pieces of metal will not discharge.

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Illumination of Mines

BY ROBERT P. BURROWS,* CLEVELAND, OHIO

(New York Meeting, February, 1916)

IN preparing this paper the object has been to set forth facts relating to illumination problems, which, judging from the results realized in the iron and steel and other industries somewhat similar to mining, will tend toward furthering safety, production, and contentment of employees, as well as economy of operation in mines. By applying the principles of illumination with the assistance of modern appliances, the full benefits in efficiency may be derived from improvements already made in other details of mine operation.

The lighting of a typical coal mine may be divided into four distinct parts: 1, The lighting of the buildings about the top; 2, the lighting of the working faces; 3, general illumination at the bottom; and 4, special applications of lighting.

The lighting of buildings about the top may be treated in the same manner as that of any other industrial plant, for we have a boiler room, an engine and generator room, a forge, a machine shop, and a hoist room. These can be well and efficiently lighted by the use of 100-watt tungsten-filament multiple lamps with proper reflectors so spaced and suspended that a power consumption of from ¼ watt per square foot in the boiler room to 1 watt per square foot in the machine shop is obtained. The methods that apply to this kind of lighting have been ably treated by a number of authors,[1] and for this reason a detailed discussion is unnecessary.

The lighting of the working faces is usually done by means of portable lamps, of which there are four types in use: The oil torch, the acetylene lamp, and the oil and the electric safety lamps. The different types have been fully described in numerous papers and articles and will not be covered here, although a few figures on the cost of operation will no doubt be

* Engineering Department, National Lamp Works of General Electric Co.

[1] *Modern Industrial Lighting.* Issued by National Electric Light Association. Industrial Lighting, *Bulletin* 20, Engineering Department, National Lamp Works of General Electric Co.

Numerous papers on the various phases of industrial lighting have appeared in the *Transactions of the Illuminating Engineering Society.*

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of interest. In attempting to obtain cost figures, one is impressed with the fact that apparently very few such data have been obtained in this country. It would seem that such data would be of particular benefit at this time, with the advent of the electric safety lamp.

The oil torch is without question the cheapest source of light. The acetylene lamp, at a cost of 6c. to 10c. per lamp per week, gives far superior illumination, but the characteristics of this source of light as well as any other open-flame lamp will bear careful consideration in view of the ever-present desire for industrial efficiency and safety. It is the opinion of many that the greater percentage of disastrous explosions in this country have resulted from the use of open-flame lamps in the so-called non-gaseous mines. This question of safety, of course, merits serious consideration.

The oil safety lamp has a distinct advantage in that it gives an indication of the presence of gas. Its development marked one of the greatest advances in mine lighting, although in most cases at the present time it is not considered a guarantee against explosion when in the presence of gas. Figures obtained from foreign countries indicate the cost of using oil safety lamps is from 7c. to 9c. per lamp per week.[2]

The electric lamp gives a steady and readily directed light, free from gases, soot, and frequent outage. A large proportion of the generated light is directed on the working face. It is sometimes considered a disadvantage that the electric safety lamp does not give an indication of gas as does the oil safety lamp. The trend of opinion in England, however, is toward choosing a lamp for the light it gives and the use of some other means for gas indication.[3] There is no question that an electric lamp passing the tests of the U. S. Bureau of Mines will give more light on the working face than any of the three previous illuminants, because it has been scientifically designed with that end in view.

Foreign practice has shown that electric light costs from 12c. to 17c. per lamp per week. This cost is about twice that of the oil safety lamp. The light on the "face," however, is materially increased by the use of the electric lamp. One foreign electric-lamp manufacturer places the cost of electric light at 2½c. per lamp per shift.[2] This figure, though it seems low, can well be realized in this country with a large installation and proper care. In this connection, it is very necessary to have proper housing and proper attention for electric lamps—more so than with the oil safety lamps. It has been found in foreign practice that this care and

[2] William Maurice: Electric Handlamps for Collieries, *The Electrician*, May 12, 1911.

F. J. Turquand: Design and Maintenance of Miner's Electric Handlamp, *London Electric Review*, March 6, 1914.

[3] *Transactions, Institution of Mining Engineers*, vol. xlix, part 1, pp. 53 to 61 (1914–1915).

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attention is very little, if any, more expensive than the attention that is given to oil safety lamps, even though more expensive help is needed, because fewer men are required to care for the electric outfits. This country has been slow in taking up the electric lamp. It has been said that in Belgium alone there are 12,000 outfits in use. The excellent work done by the U. S. Bureau of Mines to obtain the highest efficiency for this new source of illumination has accomplished what years of competition among electric mine-lamp manufacturers could hardly have brought about.

The application of the principles of industrial illumination to the general lighting of mines must be made in the face of conditions difficult to overcome. In fact, all the conditions the illuminating engineer considers most

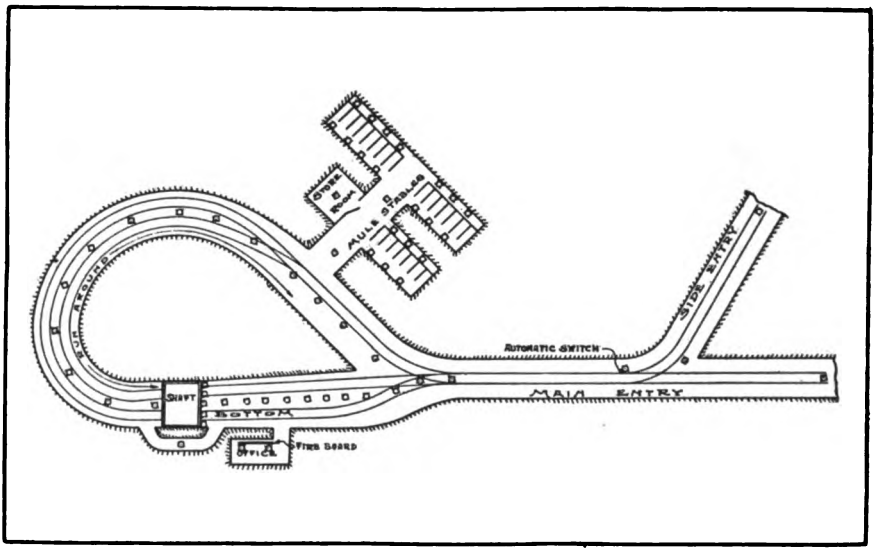

FIG. 1.—TYPICAL MINE LAYOUT.

difficult are present: low ceilings, black walls, dust, smoke, and dampness; but in spite of these, very satisfactory results have been obtained.

An ordinary mine, from a lighting standpoint, can be considered as composed of at least six parts: the bottom, the run-around, main entry, side entries, mule stables, and small rooms, such as offices, pump rooms, storage rooms, and first-aid rooms. These are shown diagrammatically in Fig. 1. The bottom, being the entry and exit for both men and coal, accommodates more traffic than any other part of the mine and should be especially considered from the standpoint of both convenience and safety. Fig. 2 shows a portion of a well-illuminated bottom and shaft opening of a typical mine. The lighting of the shaft in this case was accomplished by the use of 40-watt tungsten-filament lamps equipped with angle reflectors, placed above and across the shaft opening so as to direct the light on the

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cages. The maximum intensity is at the near edge of the cage, and the eyes of the workmen on the side of the shaft toward the observer are not subjected to the glare of the lamps. For comparison, Fig. 3 shows this same portion of the mine lighted by the use of bare carbon lamps. It is

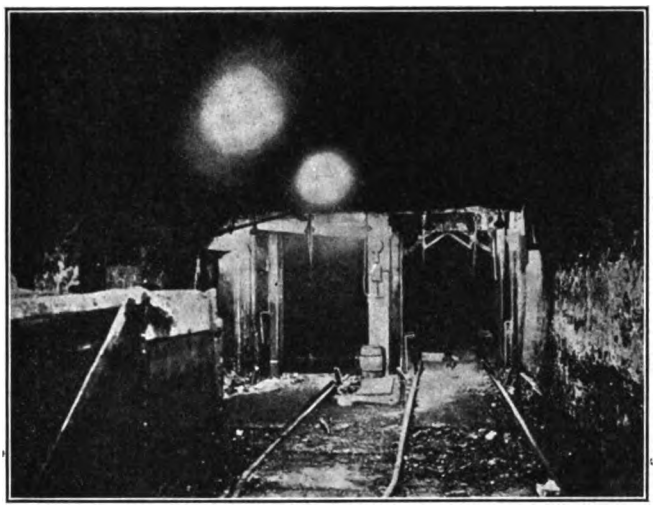

FIG. 2.—WELL-LIGHTED SHAFT BOTTOM.

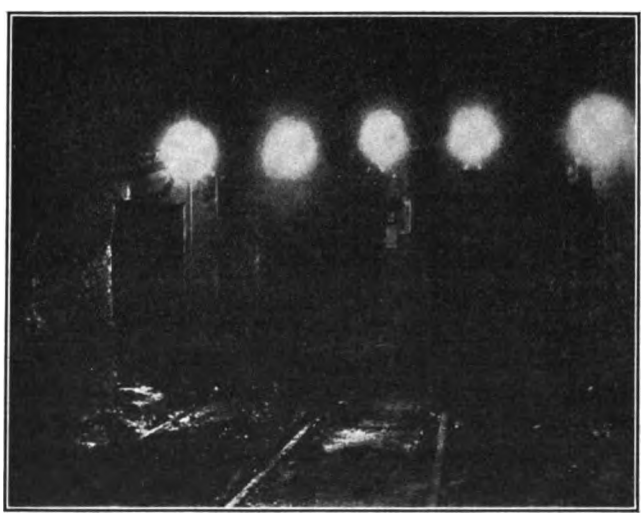

FIG. 3.—POORLY LIGHTED SHAFT BOTTOM.

readily seen that the distribution is not of the best and also that the glare of the bare lamps obscures that portion of the mine which lies beyond. These illustrations were made from actual photographs, retouched only enough to remove the halation effects of the bare lamps. The photo-

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graph shown in Fig. 2 was exposed about 1 min. as against 15 min. for the one in Fig. 3.

That portion of the bottom leading into the mine, where cars are directed on to the cages, can be well lighted with 40-watt tungsten lamps in shallow dome reflectors placed above and between the tracks. These units, spaced at about 6-ft. intervals and hung about 8 ft. above the floor, will give satisfactory distribution of light. It will be noticed from Fig. 4 that the car wheels are well illuminated and that there is practically no glare. It would be well to design the lighting of this part of the mine on a basis of 4 to 5 foot-candles at the floor, not because the work demands this intensity, but because of the greater safety which results from ample

FIG. 4.—PROPERLY ILLUMINATED ENTRY.

illumination and because dust collecting on the lamps and reflectors decreases the amount of light delivered.

The run-around should require only sufficient light to make visible any obstructions in the path of the empties as they leave the cages. This part of the mine may be illuminated with 25-watt tungsten lamps equipped with shallow dome reflectors, spaced 15 ft. apart and suspended 8 ft. above the floor. In the main entry, the function of light is not so much to illuminate as to silhouette objects which may obstruct the passageway. With silhouette lighting, a comparatively small amount of light is needed to obtain the effect desired, which is to see objects outlined against something that is lighted. For instance, whitewashed doors or walls reflecting the light toward the observer's eye are excellent backgrounds against which objects form silhouettes when in the line of vision of the observer. The glint of the light on the rails forms another good surface from which

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silhouette lighting may be obtained. With 25-watt tungsten lamps in shallow dome reflectors, spaced at intervals of about 300 ft., the height depending upon the height of the entry, the silhouette lighting is excellent. Two units, one to illuminate the switch and the junction and the other illuminating a portion of both the main and side entries, help to eliminate collisions and by the increased light warn the trip driver that his train is approaching such a junction.

The mule stables with their low roofs may be effectively lighted with 40-watt tungsten lamps equipped with angle reflectors placed along the back wall and as high as possible, one unit to each two stalls. In front of the stalls and opposite the angle units, 25-watt tungsten lamps with deep bowl reflectors may be used to illuminate the feed boxes and passageway.

The mine offices need but one 25-watt tungsten lamp equipped with a shallow dome reflector. The fireboard at the bottom should be well illuminated with one or more 25-watt lamps of this type equipped with angle reflectors, depending upon the size of the board, while the pump rooms and storage rooms may be lighted in the same manner as offices.

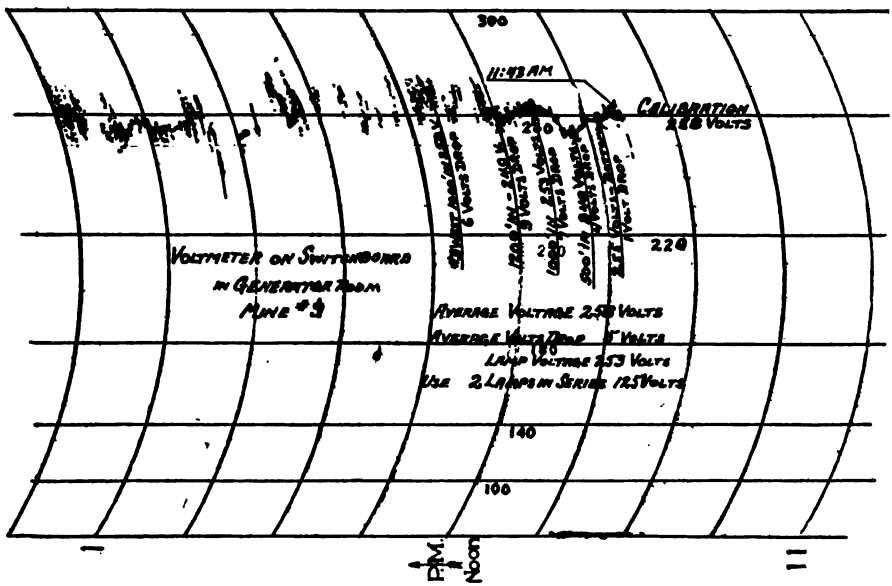

FIG. 5.—VOLTAGE RECORD OF MINE LIGHTING CIRCUIT.

The first-aid rooms, in order that the best attention be given the injured, should not only be well lighted, but should have the walls well white-washed, thereby obtaining well diffused and distributed light. Frequent whitewashing of the walls of the bottom, offices, mule stables, etc., and the walls of the entries for 20 ft. each side of the units, will greatly increase

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the illumination in these parts of the mine. Carbon lamps are most generally used in mines, but to keep the load on the generator as low as possible and maintain the most constant illumination in spite of voltage fluctuation, and to direct the light where wall and ceiling reflection cannot be relied upon, tungsten-filament lamps with weatherproof enameled reflectors will, in my opinion, be found most satisfactory.

It may be interesting, by reason of the high voltage usually found in mines, and its fluctuation, to show how the proper voltage for a lamp, to secure greatest life and light, is determined. A recording voltmeter is connected at the switchboard on the terminals of the switch controlling the lighting circuit, usually the trolley line. When this is in operation, a carefully calibrated portable voltmeter is connected in multiple with the recording meter and a section of the chart of the recording meter is compared with the readings of the calibrated portable meter. Fig. 5 shows a section of such a chart and the calibrated line. This chart should be taken over a period of at least 3 hr. and for 24 hr. if possible.

TABLE I.—*Lamp Voltages for Various Line Voltages. Street-Railway Tungsten-Filament Lamps.*

Average Line Voltages	Number of Lamps in Series	Voltage of Individual Lamp
250	2	120
260	2	125
270	2	130
280	2	135*
290	2	140*
300	3	105
325	3	105
350	3	115
400	3	130
425	4	105
450	4	110
475	4	115
500	4	120

For voltages above 500, use five lamps in series as on street-railway circuits.
* Special lamps.

Voltage readings are then taken back from the shaft along the main entry at intervals of 300 or 400 ft. by means of the portable meter, the voltage and time being recorded. A study of the chart will show the average voltage over the period taken. A comparison of the chart with the voltage readings taken back in the mine will show the average drop in the line. From the average voltage obtained from the chart should be subtracted the average line drop obtained from the readings taken in the mine, the result being the voltage on which lamps will operate to give the same life as on the fluctuating voltage in the mine.

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For average voltages up to 250 volts, regular multiple lamps should be used. For average voltages from 250 up to 280 volts, there is a choice of burning lamps in multiple or in series. The best practice is to burn two lamps, carefully selected for current, in series. Such lamps can readily be obtained and are known as street-railway lamps. For voltages above 280, the proper lamps should be selected for series burning. Table I lists lamps for specific voltages.

A recording chart, used as described previously, showed a maximum voltage variation of 20 per cent. From Fig. 6 it will be seen that with a 10 per cent. reduction in voltage, the candlepower of the carbon lamps is 20 per cent. lower than that of the tungsten lamps. Fig. 6 shows that the characteristics of the tungsten lamp are such that voltage variation does not affect candlepower as much as it does that of carbon lamps.

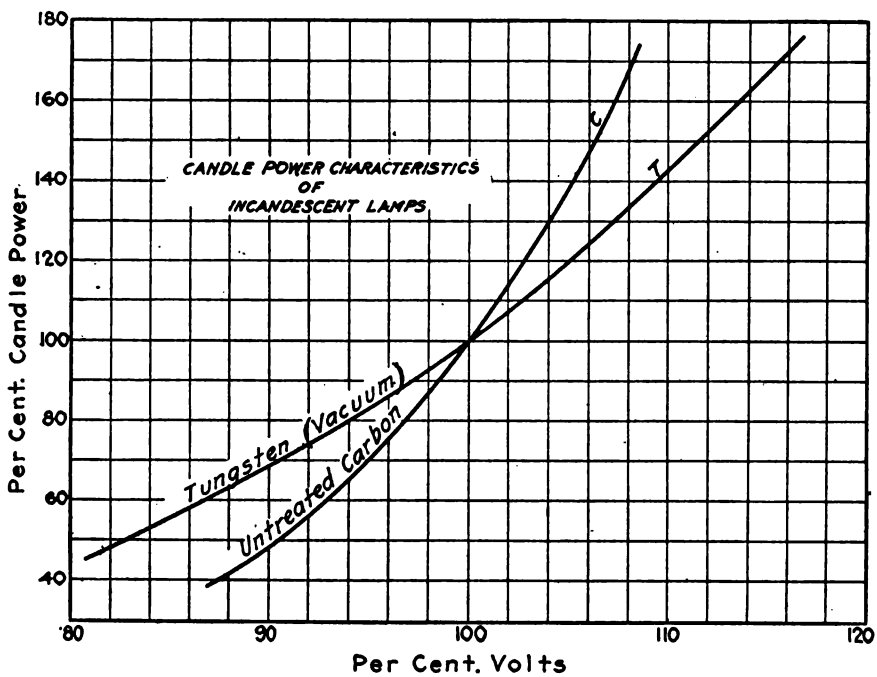

FIG. 6.—CANDLEPOWER CHARACTERISTICS OF INCANDESCENT LAMPS.

A few comparative cost figures in connection with the problem of more efficient illumination follow. Consider, for example, an installation, such as illustrated in Fig. 1, where twenty-six 40-watt tungsten lamps and reflectors and thirty-one 25-watt tungsten lamps and reflectors are to replace the same number of 32-cp. and 16-cp. carbon lamps, respectively. During a period of 300 days, at 10 hr. a day, the tungsten lamps would consume about 5,440 kw-hr., while the carbon lamps

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would consume about 14,940 kw-hr. With the cost of current at 0.5c. per kilowatt-hour, the saving in cost of power with the use of tungsten lamps would be about $50 a year. From this must be subtracted about $17 for the difference between the cost of the carbon lamps and the tungsten lamps. This will leave about $23 net saving. With the reflectors costing $60, the installation would be paid for in three years.

These figures tend to show that if dollars and cents alone were considered, it would be more profitable to use the higher efficiency lamps. This is even more marked when the illumination on the working plane is considered, because with the use of reflectors the illumination is more than double that obtained with carbon lamps.

There are many other places where special applications of lighting would tend to increase efficiency and convenience; for instance, triplights—now as a rule simply oil torches on the end of the train—could be easily replaced by small storage-battery outfits showing a red light. Locomotive headlights can be equipped with low-voltage concentrated-filament tungsten lamps in parabolic reflectors, with a decrease in trouble, increased light, and decreased breakage over the present carbon or regular tungsten filament. Two 30-volt, 100-watt tungsten-filament locomotive-headlight lamps can be burned in series with a resistance. The loss in current through the resistance is a small factor as compared with the gain in steadiness and brilliancy of illumination from the parabolic headlights. The construction of this lamp is such that maximum strength of filament is obtained, which is an essential feature where the service is as severe as on a locomotive. Another possible consideration is the placing of distinctive lights where telephones are located, or where first-aid equipment may be obtained. This could be accomplished by the use of red lights on the power circuit installed in connection with a small primary-battery system, which would operate a miniature lamp in place of the large lamp should the power circuit for any reason fail. This system has been successfully worked out in theaters where the same principle is involved.

It is hoped that, from the few figures given in this paper, it will be seen that the application of the latest scientific knowledge to the lighting of mines is not so expensive as it is generally thought to be, and should be considered as a means of increasing safety, bettering working conditions, increasing production, and at the same time decreasing the cost of operation.

Discussion

Edwin M. Chance, Wilkes-Barre, Pa.—I have been very much interested in the comprehensive and able discussion of the important topic of mine lighting just given by Mr. Burrows, especially as I know by common report Mr. Burrows' ability and his close touch with the de-

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velopment of the electric mine lamp, and its use in mines in connection with a portable electric lamp. It is regrettable that Mr. Burrows was not able to acquaint himself more closely with the cost of mine illumination, the actual cost, as produced by a portable light as carried by miners, whether a flame safety lamp or an open light. Had he been able to acquaint himself more closely with these data, he perhaps would not have arrived at some of the conclusions which he has presented to us in his paper. For example, we find Mr. Burrows advises that the open torch is by far the cheapest source of illumination, as produced by a portable light. I am very sure that is not the fact. The cost of miners' oil varies from about 25 c. a gallon to 90 c. a gallon. The various States have passed laws from time to time rendering it mandatory that the miner shall use, whether he is willing or not, a so-called better or higher grade of burning oil. These laws have entailed the use of an oil so expensive that the use of the oil torch has become practically impossible, and the acetylene or carbide miners' lamp has largely taken its place.

I have evolved a few figures showing the relative efficiency and usefulness of the open oil torch and the acetylene miners' lamp. For example, the acetylene miners' lamp consumes or burns 4 liters of acetylene per hour, or 0.14 cu. ft.; the oxygen consumed is 10 liters, or 0.35 cu. ft. per hour; air consumed, 50 liters, or 1.76 cu. ft. per hour; blackdamp produced, 40 liters, or 1.41 cu. ft. per hour; air rendered extinctive to the acetylene flame, 112 liters, or 3.95 cu. ft. per hour; carbide consumed, 13 g. per hour.

The open oil torch, on the other hand, burns about 20 g. of oil per hour; the oxygen consumed is 48 liters, or 1.69 cu. ft. per hour; air consumed, 240 liters, or 8.47 cu. ft. per hour; blackdamp produced, 191 liters, or 6.73 cu. ft. per hour; air rendered extinctive to oil flame is 1,413 liters, or 49.92 cu. ft. per hour.

In addition to this, the acetylene lamp gives a much more accurate and reliable indication of the presence of blackdamp, for the reason that the oil lamp is too sensitive to blackdamp. The latter is extinguished if the oxygen in the air falls to 17.5 per cent. A man is able to live and work with 10 per cent. of oxygen, and a miner knows, when the oil lamp is extinguished, that he can still live in the atmosphere, and he will venture into regions where the air is extinctive to his oil torch, and be overcome, whereas with the acetylene lamp, the flame of burning acetylene is extinguished in still air when the oxygen falls to 12 per cent. At 14.5 per cent. the acetylene flame loses its brilliancy, becomes long and very blue, and the miner is given an indication of the presence of blackdamp in dangerous quantities. He knows that blackdamp extinctive of his oil lamp is not so dangerous as that extinctive of his acetylene lamp and if the acetylene lamp loses its brilliancy, he knows he is near the danger limit and will not enter further into the mine.

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The illumination produced by the oil lamp has an intensity of about 2.5 cp. The illumination produced by the acetylene has an intensity of about 6, sometimes more, and sometimes as low as 5. The relative costs of these sources of illumination are about as follows: For the acetylene lamp per week of 54 hr., the depreciation charge is about 2 c. The cost of carbide is about 7.5 c., based on carbide at 5 c. per pound, and assuming a consumption of 1½ lb. of carbide per week, giving a cost per week of 9.5 c. In the case of the oil lamp, with oil at 45 c. per gallon, assuming that the lamp consumes 0.33 gal. per week, which has been found to be the average figure, this is a little low; the cost for oil is 13.6 c. per week. The cost of about one-half ball of wick cotton, which will be used in the wick during a week, is 2.5 c., making the total cost per week for the oil lamp 16.1 c., an increase over the acetylene lamp of 6.6 c. per week.

In addition to this, the cost for the acetylene lamp per week-candlepower is 1.6 c., whereas the cost for the oil lamp is 6.4c. per week-candlepower, showing clearly that the relative cost of the acetylene lamp and oil lamp is greatly in favor of the acetylene lamp.

This has been borne out in practice. I have yet to see the men of a colliery that has abandoned the use of the oil lamp for the acetylene lamp willing to return to the oil torch. In addition to this, the oil torch, because of the nature of the materials burned, produces a large amount of sticky soot and more or less offensive odor, which have been found to be very trying to the miners.

In regard to the use of the so-called electric safety lamp in mines, I will say that this usage is at present in its infancy. I believe in the very near future we will have mine electric lamps that are very much more satisfactory than those which have been used in the past.

There is no question but that this art is making rapid strides, as the author of the paper has so well stated, and that the Bureau of Mines has been especially active in promoting the improvement of these lamps.

Regarding the relative cost of the electric lamp and the oil safety lamp, I will not go into detail for the reason that the oil safety lamp is provided by the coal operator, as required by law, remains his property, and is maintained by him.

T. M. Chance, Wilkes-Barre, Pa.—(Demonstration with lamps).—I think I could not add anything in direct discussion of this paper, but the members of the Institute may be interested in some work we have done in attempting to produce an improved safety lamp.

The great difficulty experienced in safety-lamp lighting is the low illumination available in the present forms of oil-burning safety lamps. The candlepower of these lamps will vary from 0.15 for the original Davy lamp up to a maximum of about 1.60 for the latest type of Ackroyd and

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Best kerosene-burning lamp, which is obtained by the use of a reflector, the mean horizontal candlepower of this latter lamp being between 1.00 and 1.20 Standard English Sperm candlepower. While somewhat higher values than those quoted have been obtained with the Davy lamp, we have found the figures given to be about the average attained in practice (see Tables I and II).

TABLE I.—*Photometer Tests**

Standard = 1.03 Standard English Sperm Candle—All readings corrected to Unit Candlepower

Lamp No.	Fuel	Time Lighted	Time Tested	Candlepower Average of Four Readings		
				End	Flat	45°
1	Sperm oil	10:30	11:30	0.115	0.115	0.115
2	300° kerosene	10:30	12:00	1.10[c]	1.58[a]	1.12[b]
3	Benzine	10:30	12:10	0.69	0.69	0.69
4	Benzine[d]	10:30	12:20	0.83	0.83	0.83
5	Acetylene	11:00	12:25	0.97	0.97	.....
6	Acetylene	11:00	12:35	3.20	3.20	.....
7	Acetylene	3:00	4:00	4.00	4.00	.....
8	Acetylene	3:00	4:30	5.50[a]	3.20[c]	3.60[b]
9	Acetylene	3:20	4:20	5.60[c]	17.00[a] 5.75[c]	6.60[b]
10	Acetylene	1:00	1:35	16.50[a]	3.35[c]	4.10[b]
11	Miner's oil	1:00	1:40	.....	1.54	.....

[a] Reflector in use.
[b] Reflector partly in use.
[c] Reflector not in use.
[d] Lamp smoking and flame in gauze.

The highest illumination that has hitherto been available is that reached in the Ackroyd and Best lamp. This lamp, which is made in England, burns 300° kerosene and gives a white flame of fairly intense illuminating power. The lamp is of the Muessler type with a small glass chimney fitted to the lower end of the Standard Muessler chimney, and completely surrounding the flame. This construction greatly improves the ventilation of the flame and the increased illumination is largely due to this improved ventilation. The principal defect of the lamp is that heavy jars, or falls, extinguish it, and to permit its efficient use it has been found necessary to install electrical reigniting stations underground.

One of the best types of liquid-fuel burning safety lamps used in this country is the benzine lamp. The Wolf lamp, built in Europe, and used generally in this country, is probably the most familiar type of benzine

* Made by E. M. Chance, Wilkes-Barre, Pa.—Mar. 15, 1916.

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lamp in use. This lamp is of the Marsaut type with bottom air feed and possesses admirable qualities. The illuminating power of the lamp has been claimed to exceed 1.0 candlepower, but in our tests we have never been able to get so high a result. This lamp has the same defect, although to a somewhat less degree, as the Ackroyd and Best, *i.e.*, ease of extinction. The Wolf people have overcome this trouble by the use of friction reigniters, either of the phosphorous match or Auer Metal type, a method of reignition of course impossible to use with a non-volatile fuel such as kerosene.

TABLE II.—*Description of Lamps Tested in Table No.* I

Lamp No.	Description	Type	Wick	Burner
1	(Hughes Bros.) Davy	New Castle-Davy	Round	Tin
2	Ackroyd & Best safety with reflector	Muessler with glass chimney	Flat	Porcelain
3	Wolf safety	Marsaut-bottom air feed	Round	Brass
4	Wolf safety	Marsaut-bottom air feed	Round smoking	Brass
5	5-hr. Wolf safety	Marsaut-bottom air feed	.....	2 No. 1 lava-tip Baldwin flames, ⅜ in. high.
6	12-hr. Wolf safety	Marsaut-bottom air feed	.....	2 No. 0 Baldwin, lava-tip
7	12-hr. Wolf safety	Marsaut-bottom air feed	.....	2 No. 4 Baldwin, lava-tips
8	12-hr. Wolf generator, Ackroyd & Best top & reflector	Muessler with glass chimney	.....	2 No. 4 Baldwin, lava-tip
9	Baldwin half-shift open-hand lamp	ITP feed	.....	Baldwin, fish-tail lava-tip
10	Baldwin zar	Cap lamp	.....	Baldwin, lava-tip
11		Standard cap lamp	Round	Tin

This use of internal reigniters has been the subject of much discussion. I understand that in West Virginia such igniters must be so made that the individual miner cannot relight his lamp but must apply to a mine boss or other official provided with a key for this purpose, and I believe a somewhat similar regulation has been proposed in England, as it has been thought that some otherwise unexplained explosions have been produced by careless reignition of lamps provided with such reigniters.

In endeavoring to produce a safety lamp of improved illuminating power we felt that the liquid-fuel burning lamps already develop about the maximum illumination that can be secured with these fuels, and as the candlepower of the portable electric lamp cannot be increased

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without a corresponding increase in the weight of the battery, it seemed to us that the most promising field for improvement was in the development of a satisfactory acetylene lamp.

A number of European investigators have attempted to produce a practicable acetylene safety lamp but these attempts have failed because of the ease with which the acetylene flame is extinguished by concussions from shot firing. These investigators attempted to overcome this

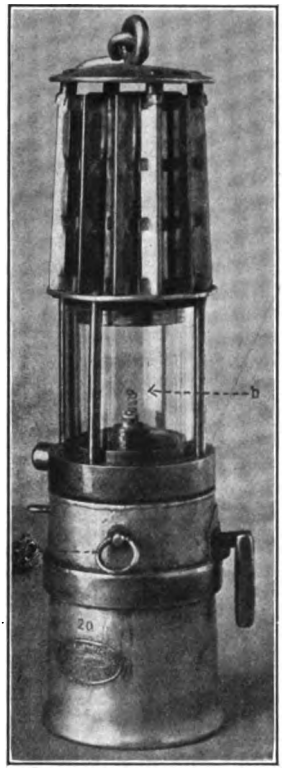

FIG. 1.—ACETYLENE SAFETY LAMP.

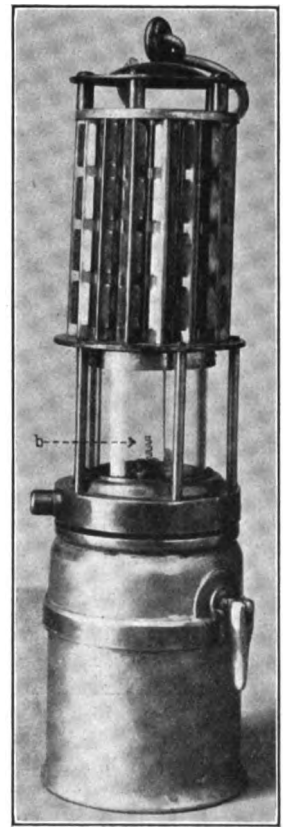

FIG. 2.

difficulty by the use of internal reigniters similar to those applied to the benzine lamps of the Wolf type, but such a method of reignition removes the lamp from the safety lamp class, because with manually operated reigniters of this kind, internal explosions may be produced in the lamp that will not only pass flame through the gauzes, but may even wreck the top of the lamp. This is due to the formation of explosive mixtures of acetylene and air in the time interval between the extinction of the flame and its reignition by the miner. The Belgian investigators at

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Mons made elaborate tests of acetylene safety lamps fitted with reigniters of this type and as a result of these tests the use of acetylene lamps with internal manually operated reigniter has been prohibited in Belgium.

It seemed to us that this difficulty could be overcome if a reigniter could be made that was automatic and instantaneous in action. This conception of the necessity for an automatic reigniter resulted in the production of the lamps shown in Figs. 1, 2, and 3. These lamps are Standard 5- and 12-hr. Wolf acetylene safety lamps in which the manually operated reigniter, noted by the ring "a" in Fig. 1 has been replaced by the automatic reigniter "b." This automatic reigniter is in this case formed of a flattened coil of No. 20 nicrome (nickel-chromium

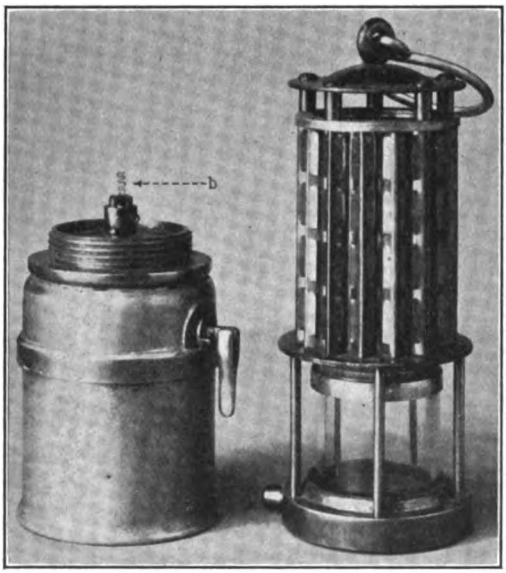

FIG. 3.

alloy) wire, placed between the flames of a two-jet burner. In operation the reigniter is heated by radiation from the flames, no portion of it being in contact with either flame. Immediately following the extinction of the lamp by concussion, the reigniter (which is red-hot) relights the jets as soon as the flow of acetylene through the burner is reëstablished. There is no time interval between the extinction of the flame and its reignition during which an explosive mixture could form within the body of the lamp, hence internal explosions are impossible.

The lamp shown in Fig. 2 was subjected to the effect of a dynamite shot, lifting bottom, in one of the mines of the Wyoming region. This was a most severe test, the lamp being placed within 30 ft. of the shot. Several open acetylene lamps were placed beside it, and other open

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acetylene lamps were with the observers at a point 60 ft. up the gangway from the safety lamp. All these lamps except that fitted with the reigniter were instantly, and of course permanently, extinguished.

The acetylene safety lamp cannot be extinguished by shocks or jars as in the case of the benzine or kerosene burning lamp, and the automatic reigniter which we have provided eliminates the question of extinction from concussion. We believe that the use of this lamp will effect increased efficiency in the operation of safety-lamp sections underground.

E. M. CHANCE.—What candlepower has this acetylene lamp?

T. M. CHANCE.—The 5-hr. lamp, with the flames rather high has a candlepower of 3.0, and in the 12-hr. lamp we normally get 3 to 4 candlepower.

S. A. TAYLOR, Pittsburgh, Pa.—What is the price of this compared with the other?

T. M. CHANCE.—That is a commercial question.

S. A. TAYLOR.—It is commercial, but still I think some of these men would like to know.

T. M. CHANCE.—I am speaking as an engineer, and not as a manufacturer, but of course I am interested in the matter of price and I believe the manufacturers will be able to put these lamps on the market to sell at about $3 to $3.50.

GEORGE RICE, Washington, D. C.—What is the method of relighting after the stored heat is dissipated?

T. M. CHANCE.—There is none at all, and I think it would be a fatal objection if we had such a thing. If the lamp does not generate properly, and due to that is extinguished, I think the miner should not be allowed to tamper with it. If the lamp is charged with carbide and water and fitted with a properly designed feeding device, it must continue to burn unless extinguished by concussion, and that is taken care of with the automatic reigniter. When the lamp is upset, the flame commences to burn in the gauzes (due to insufficient oxygen supply) but reëstablishes itself at the burner when the lamp is righted.

GEORGE S. RICE.—Don't you find that with an explosive mixture of acetylene gas and air, the flame will pass through the gauze, whereas the flame of a methane mixture will not?

T. M. CHANCE.—I presume you are alluding to the Belgian experiments. These experiments showed that a lamp generates acetylene with the flame extinguished, and that this acetylene may be fired by a burning lamp, provided you have these conditions: A lamp hanging

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perfectly motionless, in an absolutely quiet atmosphere, with a perfectly explosive mixture of acetylene and air. I believe M. Lemaire, who was in charge of this work, made the statement that while this was true, it was not fair to the lamp, because in a mine the lamps are not used hanging in a tightly closed box. The best proof of this can be shown by placing the two lamps together, one with the flame burning and the other generating acetylene with the flames extinguished. It is readily seen by placing the burning lamp side by side with the lamp generating acetylene that no passage of the flame takes place to the exterior acetylene.

GEORGE S. RICE.—Then your system to meet that particular condition would be to have underground relighting stations or stations where the miner could change his unlighted for a lighted lamp?

T. M. CHANCE.—Just as in any colliery where a large number of safety lamps are used and where the percentage of outage is large. I understand that in England at one colliery test an average of 2.5 per cent. of the lamps used underground were out per shift. As I said before, the only thing that can extinguish this lamp is failure of the generator to generate acetylene. I do not think a lamp can be produced that will be absolutely fool-proof. You cannot build an electric lamp in which the wiring cannot be short-circuited or the battery cannot fail.

D. B. REGER, Morgantown, W. Va.—Have you ever tried any different mechanical arrangement of the safety lamp so that you can get different illumination in the roof of the mine and the bottom of the mine? My experience was that with the safety lamp I could not see the roof at the top nor see the bottom of the mine.

T. M. CHANCE.—You can see the roof with an acetylene lamp much more clearly than with the other types; the roof looks gray instead of black. I believe there might be some advantage in producing a lamp in which some of the light was reflected from the top. That is merely a matter of optics.

R. V. NORRIS.—Is it a question of utility?

T. M. CHANCE.—And optics to get a satisfactory lamp.

R. V. NORRIS.—Can that lamp be put out?

T. M. CHANCE.—Yes, it has a valve for that purpose.

R. V. NORRIS.—How?

T. M. CHANCE.—By turning the valve.

R. V. NORRIS.—Are you now generating acetylene under pressure?

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T. M. CHANCE.—No, the water is cut off and the gas valve is shut and what little generation is still going on is bypassed to the atmosphere.

GEORGE S. RICE.—This appears to be an interesting and perhaps a very valuable discovery. Do you think there is any material danger, if two men are provided with a lamp, and one lamp has gone out, and the men are carrying the lamps in juxtaposition, of gas generated from the first lamp being ignited in the second lamp, the flame flashing through the gauze?

T. M. CHANCE.—Absolutely none, because if the men are walking together they are moving in the air.

As to this other question of the firing of one lamp from another, it is simply a matter of refrigeration, *i.e.*, of having sufficient screen capacity. This can be seen by lighting the acetylene generated upon the top of the gauze. The acetylene remains ignited and burns on the exterior of the gauze like a Bunsen burner, but the flame does not pass through the gauze into the body of the lamp. The gauze must be in a red-hot condition before it will pass the acetylene flame. I would like to emphasize the fact that acetylene has no peculiar characteristics of importance in this connection, other than its low kindling point.

E. M. CHANCE.—These lamps generate about 5 or 6 liters of acetylene an hour, and the volume of explosive mixture is so insignificant that it is almost impossible to obtain an aura about the lamp, an explosive aura, when there is the minimum movement of air. The acetylene is generated at a higher temperature than the air, and tends to rise and move away from the lamp all the time.

E. T. LEDNUM, Chicago, Ill.—You spoke of an experiment with an explosion of dynamite, or high explosive, as extinguishing that light. Have you ever tried it with black powder?

T. M. CHANCE.—The small 5-hr. lamp with a single-jet burner was tried on two different occasions and successfully withstood the effect of a black powder shot at both places.

E. T. LEDNUM.—The black powder shot is of longer duration.

T. M. CHANCE.—That may be true, but it does not have the same reversal of pressure that accompanies a dynamite explosion, and it has nothing like the effect on an acetylene lamp.

R. DAWSON HALL, New York, N. Y.—What are the weights of the various lamps?

T. M. CHANCE.—The 12-hr. lamp weighs 5 lb. 13 oz., charged with carbide and water; small lamp 3 lb. 13 oz.; Wolf benzine lamp 3 lb. 8

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oz.; Ackroyd and Best 3 lb. 8 oz.; small Davy lamp 2 lb. 4 oz.; 12-hr. acetylene lamp is massively built, and we think the generator can be considerably lightened.

R. DAWSON HALL.—What possibilities are there in the acetylene lamp of breaking the glass of the lamp; that seems to be the weakest point in it; when you lean it over on one side you are liable to break the glass.

T. M. CHANCE.—There is this difference: When the oil lamp is upset, if the light does not go out the flame burns against the glass and is sure to break it, whereas with the acetylene lamp the flame almost immediately commences to burn up in the gauze and thus saves the glass. The Belgian Commission made a number of tests of glass breakage with acetylene lamps, and did not find it a serious or valid objection to the lamp.

R. DAWSON HALL.—Have you determined as to the "pick-up" power of the lamps, the amount of fire-damp you can pick up with the lamps?

T. M. CHANCE.—That is simply a question of heat value; if you have a hot flame you can pick up the gas. The European experimenters have found no difficulty in that direction.

HUGH ARCHBOLD, Scranton, Pa.—Some time ago I stuffed gauze on the ordinary acetylene cap lamp and tested out in competition with the Davy, to find out if the acetylene picked up as well. With the long flame that the acetylene lamp gave, I found I was able to pick up a cap where I could not do it with the Davy.

T. M. CHANCE.—About 1840 the Muessler lamp was introduced, and since that time the vast majority of safety lamps used abroad, and in later years in this country, have been lamps equipped with a glass shield over the flame, and the experience of almost 80 years has shown that the danger due to glass breakage is a negligible quantity. When a glass breaks in a well-designed safety lamp the upper portion of the lamp structure holds it in position, and as it is placed under compression when the lamp is screwed together it stays in position in the lamp body.

The miner's electric cap lamp is a very different proposition; if the miner does not catch his battery against the caps in an entry he is exceedingly likely to strike the roof with his head and break the lamp glass. The strength of the average cap lamp protective glass and electric bulb is very different from that of the cylindrical safety lamp glass held tightly in compression.

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G. H. STICKNEY, Cleveland, O.—Referring to the lamp on exhibition, I would like to inquire about the possibility of gas ignition if the glass is accidentally broken. This point was very strongly emphasized by the Bureau of Mines in connection with the design of electric mine lamps.

No doubt the members of the Institute have an immediate interest in the particular apparatus described here. As an outsider, however, I am disappointed that the discussion has not taken up the general subject of mine lighting suggested by the paper. I sincerely believe that the provision of better illumination in mines will, in the long run, prove of vastly more importance. In the past few years one industry after another has emerged, so to speak, from the gloom, and whereas formerly it was considered practicable to get along with almost no light at all, higher and higher standards have been demanded. These have resulted in remarkable gains in safety, efficiency, and economy. While mining experts today seem to feel they are getting along fairly well with the low intensity employed, I believe before long it will be found not only profitable, but actually necessary, to raise the standards of illumination far above any which you now conceive.

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Broken Hill Underground Mining Methods

BY E. J. HORWOOD,* BROKEN HILL, N. S. W., AUSTRALIA.

(New York Meeting, February, 1916)

THE varying physical character and large extent of the Broken Hill lode necessarily involve the employment of a variety of underground methods. The lode had its origin in an extensive fault plane traversing metamorphosed schists conformably, as a rule, with their beds of stratification. The underground waters carrying minerals in solution deposited their contents in the original cavities formed by the faulting action, and in the enlargements of these cavities due to dynamic forces brought to bear on the rocks, more especially on the hanging-wall side of the fault. This deposition was supplemented by metasomatic replacement of a portion of the original rock contents by the argentiferous sulphides of lead and zinc which form the staple products of the district.

Although the orebody is practically continuous throughout the mines, its width varies greatly, ranging from a few feet to about 350 ft. The widest portions occur in conjunction with huge folds in the inclosing country rock, almost exclusively on the hanging-wall side. The ore in these folds pitches to the south in the southern half of the field, and to the north in the northern half; there are, however, undulations in these ore channels evidently due to compression of the rocks in the direction of the channels. In the earlier days, before the orebodies had been opened up, vertical cross-sections across these bulges in the hanging wall gave the lode the appearance of the "saddle formation," so well exemplified at Bendigo in Victoria; but subsequent development of the ore channels has long since proved that the formations in the two districts are entirely different.

The depth reached by the zone of complete oxidation of the sulphides varies from about 250 to 550 ft. from the original outcrop, while partial oxidation extends in places below the 1,000-ft. level. The result of oxidizing influences has been the production of ore of every grade of cohesiveness, from that of dry sand to that of hard compact rock.

The methods of mining followed in this field, and even in individual mines, are varied in accordance with the character of the ore. The general practice of the field is to sink vertical shafts, generally on the foot-

* Manager, Broken Hill Proprietary Co., Ltd.

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wall side, free from the liability of disturbance from settlement due to stoping.

Where there is an assurance of depth of orebody, the levels are spaced at distances up to 200 ft., which is considered the maximum for economical working, for, although the cost of opening up each level per ton of ore commanded would be decreased by a further increase in the spacing, the extra costs due to extra wear and tear of chutes, reduced accessibility of the stopes, etc., would more than counterbalance that saving.

Shaft Sinking

The largest shafts in the district measure 13 ft. 8 in. by 9 ft. 6 in. within timbers, and are divided into three compartments—two for winding purposes and one for ladderway, and to accommodate air main, pump column, electric light and power cables, etc. The details of the shaft timbering are shown in Fig. 1. Each winding compartment carries cages capable of holding two ore trucks, end on, each about 25 cwt. capacity; draft horses are also sent up and down in these cages at the end of each shift.

Bearers, the ends of which are let into hitches cut in the solid rock, are placed about 50 ft. apart vertically, and below them, in addition to wood blocking, each wall plate is hung from that above by means of wrought-iron hangers, as illustrated, until the weight can be taken by the succeeding bearers. Strong frames are usually hung below the lowest wall plate to protect it from flying rocks, the result of blasting operations when sinking.

The maximum distance of the timbers from the shaft bottom during sinking varies according to the nature of the ground. The general practice in sinking these shafts is to employ four reciprocating rock drills on two stretcher bars, and to arrange the holes so as to be able first to fire out a central cut across the shaft, after which the holes adjoining the cut are fired in succession by using varying lengths of fuses. A depth of about 6 ft. is gained with each firing. Electric firing has been tried on various occasions, but was not adopted because of the greater economy in explosives resulting from the use of time fuses, which enable the burden on the various holes to be successively reduced.

In sinking Delprat shaft on the Proprietary mine from the surface, the bottom of the shaft was illuminated by rays of electric light thrown down the center by means of a parabolic reflector and a mirror placed on the surface at an angle of 45°. In order to avoid shadows, the dividing timbers were left out until the sinking was completed. Operations were thus materially facilitated, especially as there was a considerable quantity of dripping water. Powerful first-motion engines and buckets of 20 cu. ft. capacity were used for sinking direct from the

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surface, but in deepening existing shafts considerations of available room tend to restrict the size of the hoists, which, in such cases, are usually geared. As these lifts, however, are smaller, rapid hoisting is not so important. Travelers, whose depth, to prevent jamming, is not less than twice their width, are used to guide the bucket through the timbering; these work in ordinary runners.

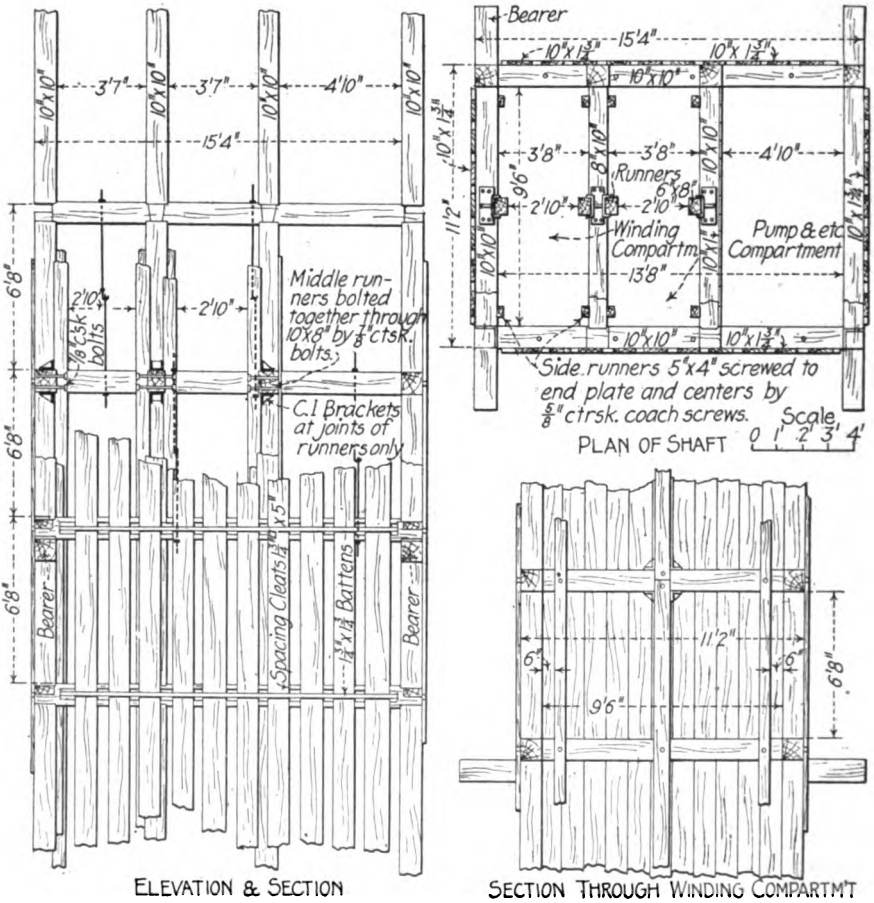

FIG. 1.—PLAN OF DELPRAT SHAFT WITH DETAILS OF TIMBERS, BROKEN HILL PROPRIETARY CO., LTD.

The Kinetore shaft on the Central mine was recently sunk for a time with the use of hammer drills held in the hand, the maximum depth of hole being from 1.5 to 2 ft. The system was, however, not persisted in, because with the particular type of drill used the effect of the vibration was too severe on the men using the machines continuously for the considerable periods necessary, and, in addition, the substitution of short lifts for the deeper sinks to which they were accustomed was not favored

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by the contract miners. So the reciprocating drills were finally reverted to. There seems, however, to be every probability that shaft sinking may be advantageously carried out by this method with machines having less vibration, and further attempts will no doubt be made in that direction.

Where new shafts are being provided to command existing workings, the practice is to put up rises about 6 ft. square from the various available levels, subsequently stripping the sides to accommodate the shaft timbering. Rising is effected with the modern air-fed hammer drills using water, either through hollow steel or by means of sprays, to allay the dust. By this method shoveling is avoided; the expenditure in explosives is reduced; delays caused by baling water are eliminated; and power is saved in hoisting, since the broken rock can generally be used for stope filling on the various levels.

Winding Engines and Equipment

The general practice of the field is to employ first-motion, double-drum steam engines, fitted with auxiliary steam cylinders controlling the brake release and reversing gear, and thus reducing to a minimum the physical exertion of the drivers. The function of the auxiliary controlling the brakes is to keep the brakes off, as counter-weights supply the power for the brakes, which would be automatically applied in the event of failure of the steam appliances. Post or pillar brakes are generally used with Ferodo linings, those having parallel motion giving the best results. Piston valves are generally used on these engines, the cutoff being generally at a fixed position and full steam applied until the cage is within a safe distance of the surface, when the steam is turned off; but in the case of the South mine more elaborate expansion gear is employed with satisfactory results.

The exhaust steam from the principal winding engines of the district is discharged into accumulators feeding the exhaust-steam turbines which furnish electric power for general purposes.

Revolving indicators enable the driver to watch the position of the cages, and in the case of Delprat shaft on the Proprietary mine the view of the brace, etc., is obstructed so that the driver's attention shall not be diverted, thus requiring him to work only according to his signals and indicators.

On the larger engines plough-steel wire ropes of Lang or Albert lay are used, 1¼ in. in diameter with a factor of safety of about 10 or 12 on dead pull when new, and having six strands of 17 or 19 wires, the former where the pulleys and drums are of ample diameter.

On the Proprietary mine springs are used, as shown in Fig. 2, near the attachment of the ropes to the cages to take up the shock when the

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weight is coming on to the rope with a view to increasing the useful life of the ropes.

Regarding safety appliances to guard against overwinding and breakage of ropes, in addition to those on the cages (on which grips engaging the sides of the timber runners are favored), safety hooks of the well-known Ormerod type (see Fig. 3) are used, which disengage the rope on passing through thimbles placed below the pit-head pulleys and support

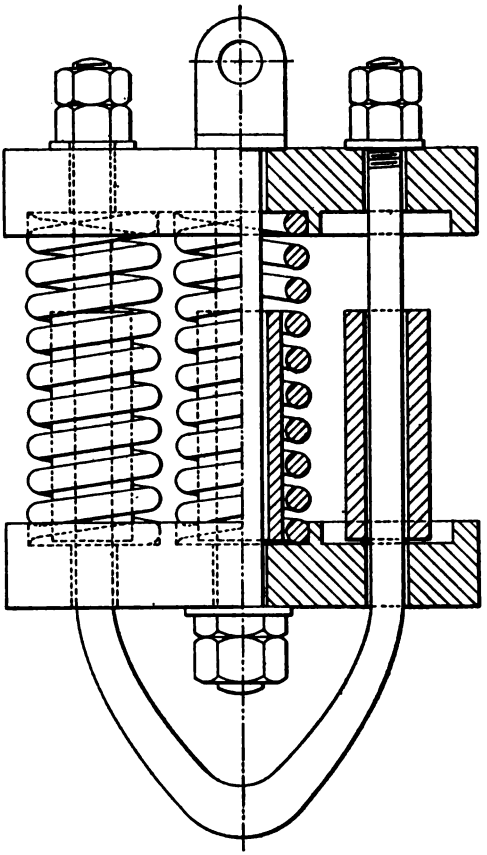

FIG. 2.—SPRINGS FOR REDUCING SHOCKS TO WINDING ROPES.

the cage. In addition, safety chairs are fixed above the top brace to catch the bottom of the cage if the other appliances fail.

At all shaft entrances lattice-work iron doors, balanced, are provided to prevent persons entering the shaft, and in addition heavy bars of steel rail, also balanced, are used to prevent trucks entering the shaft.

Poppet heads of the gallows-frame type are in general use, constructed of steel lattice framing, built clear of the engine house, which is generally of masonry.

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The equipment of the top brace, or landing, includes a revolving tippler on to which a truck of ore can be tipped by an ore inspector whose duty is to bring to the notice of the underground manager any cases found where mullock in undue quantity, or pieces of ore too large for the breakers, are included, in which cases offenders are punished by suspension or dismissal, according to circumstances.

As a prevention against fire being communicated from the surface

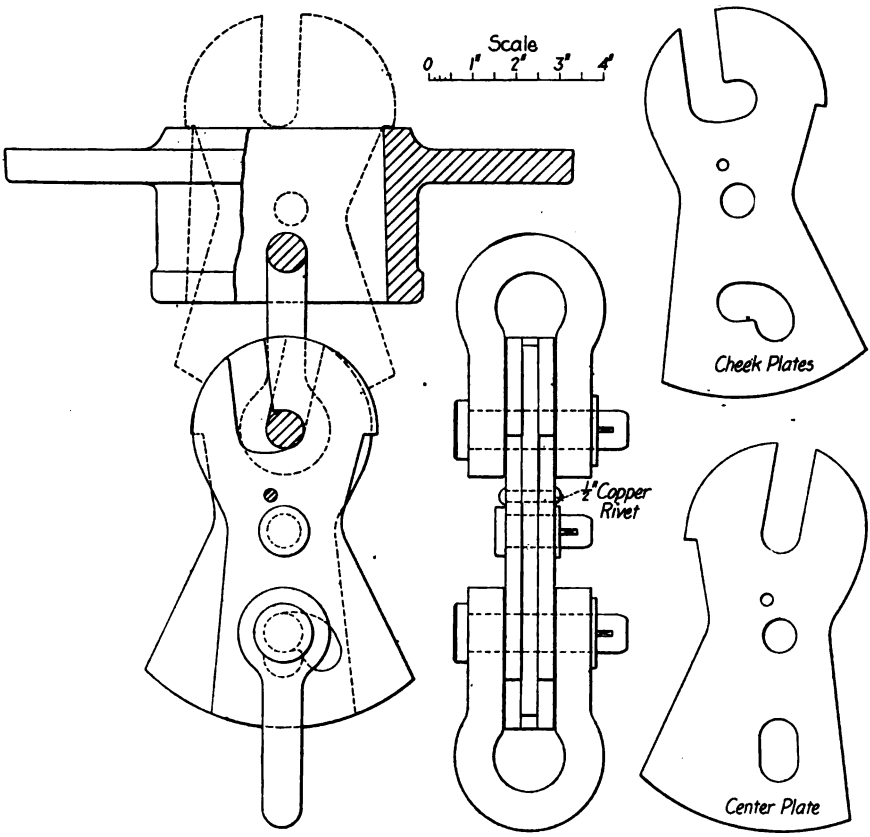

FIG. 3.—ORMEROD TYPE OF SAFETY DETACHING LINKS.

works underground, or *vice versa*, a section of the shaft timbering on the Proprietary mine is cut out just below the surface chairs; heavy plate-iron doors are hinged to the sides and can be closed at short notice. Like all the safety appliances, these are thoroughly tested periodically to insure their efficiency when needed. In conjunction with these, plate-iron doors are provided at all approaches to the shafts to control the air currents in case of fire.

In only one instance in the district, namely, at the Zinc Corporation,

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is the system of skips with ore pockets below the plats used. It is found generally more convenient, in view of the moderate tonnages hoisted at individual main shafts, to use cages, which are then always available for use in hoisting and lowering men, horses, and material as well as the ore trucks. Moreover, the latter system enables the weight

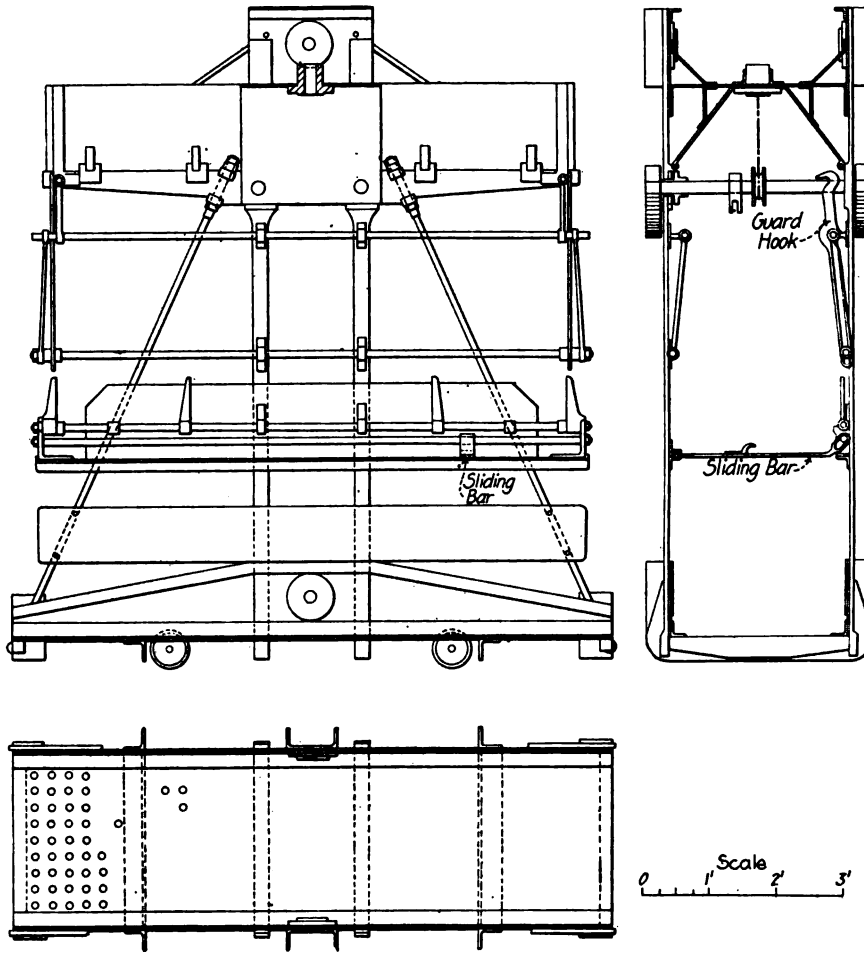

FIG. 4.—CAGE ACCOMMODATING TWO ORE TRUCKS.

of each party's trucks of ore from any number of levels to be definitely ascertained at the single weigh-bridge on the surface.

In the case of the Zinc Corporation, owing to the regularity of the orebody and (in the upper levels) its moderate width, it was practicable to pay contractors by measurement, so that the disadvantage last mentioned as attaching to the use of ore skips does not exist.

At Delprat shaft on the Proprietary mine, where as many as 1,650

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trucks of ore have been hoisted from about six levels in an 8-hr. shift, after changing half the number of men employed on the shift and with the usual 20 min. for crib, a back-shunt is provided at the back of each plat, into which the empty trucks are pushed; from the end of this back-shunt the trucks automatically gravitate to a line parallel with the full lines and laid on the requisite grade, and of sufficient length to accommodate 25 or 30 empties, which are taken away by horse traction in "rakes" of 10.

The use of cages (Fig. 4) capable of carrying two trucks, end on, facilitates rapid caging, and this is further assisted by the use of a sliding bar which retains the empty trucks in the cage on the loading side of the shaft, and *vice versa* as regards the loaded trucks, rendering it unnecessary to handle more than one bridle at each caging or uncaging of trucks, as the sliding bar is automatically pushed to and fro into position by the moving trucks.

For horse traction, the general practice is to use trucks with wheels fast on the axles; but where the distance of the shaft from the ore chutes is short, hand-trucking is practiced, and in some of these instances one wheel only on each axle is fast, facilitating trucking on curves.

At the South mine a system of automatic signals has been introduced for the purpose of indicating to the engine-driver whether the chairs are in or out for each level; this is achieved by the use of electric contacts attached to the chairs and a series of electric lights in front of the driver in the engine room. Although this device is claimed to have proved itself reliable, the managers of other mines, where the shafts are probably wetter and short-circuiting is, therefore, more likely to occur, have preferred to guard against chairs being left in by requiring the driver, in changing over to a new level, to lower the cage slowly when passing levels for the first time.

Pumping Facilities at Main Shafts

As the various levels are laid out on a gradient of about 1 in 190, or steeper, toward the shaft, the drainage of the mines is conveniently concentrated at the shaft plats, where excavated tanks are provided below the rail level at the pumping stations to hold sufficient water as sumps. Electrically driven geared pumps of various makes are used, pumping against heads of from 400 to 1,000 ft.

Communication between Surface and Underground

Telephonic communication between the surface and the various levels and isolated portions of the principal mines is established by connection of these places with the surface mine exchange. Hoisting signals are given by means of balanced wire ropes, the moderate depths of the local mines permitting the satisfactory use of this means of communication.

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Ventilation

Artificial ventilation is generally practiced on the field, fans of the Capel type and of about 70,000 cu. ft. per min. capacity against 3-in. water gage being most common. Other types of similar capacity represented here are the Waddell and the Sirocco. On the Proprietary mine, which covers ¾ mile, are both a Capel and a Waddell fan. In general

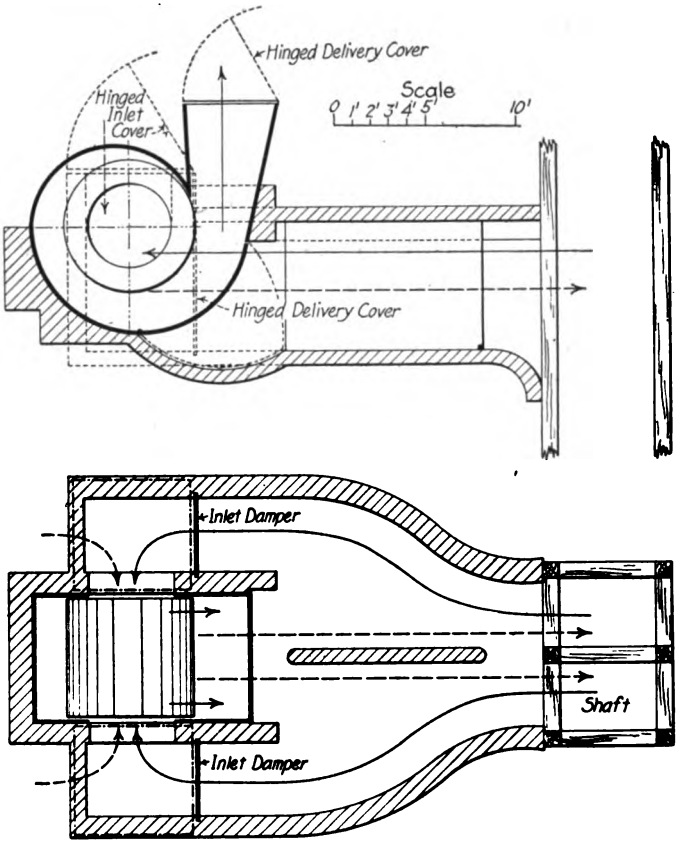

FIG. 5.—ARRANGEMENT OF AIR DUCTS, DOORS, ETC., FOR REVERSING VENTILATING CURRENTS.

use these fans exhaust the air but ducts and doors are provided, as shown in Fig. 5, by which these fans can be promptly made to act as pressure fans in case of fire making a change of draft advisable.

General

Magazines capable of holding sufficient explosives for 24 hr. are established on each important level, and generally, in conjunction with

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them, stores of tools are provided, the rule being that no new article will be issued before the return of the old. All tools, except drills and picks, are branded with numbers for identification. Explosives are issued to contractors and charged against their contract, at the expiration of which a statement of earnings and all details is furnished to them. The majority of the contracts are for 4 weeks; but in some cases where the working conditions are liable to change considerably during the period of contract, they are for 2 weeks only. In some other cases, where conditions remain unchanged, contracts are sometimes let for several months.

Tools are provided free of charge, but any loss sustained other than through fair wear and tear is chargeable to the contractors, who constitute the great bulk of the workmen. Wages-work was general in stopes up to 1892, but the efficiency of the work fell so low that contract, or piecework, was insisted upon, and generally introduced after a strike lasting 18 weeks. Owing to the higher earnings prevailing under this system, the majority of the men now favor contract work, which, in view of the inherent difficulties of supervising wages-men in so many isolated localities, is the only practicable means of securing efficiency.

Underground Development and Extraction Levels

The general practice of the field, except where the lode is narrow, is to place the extraction levels in settled country, at a safe distance from

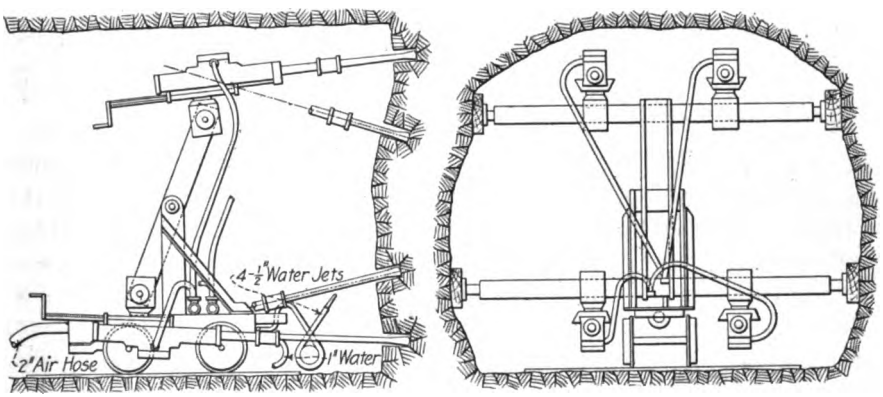

FIG. 6.—ROCK-DRILL CARRIAGE USED AT CENTRAL MINE.

the lode on the foot-wall side. These are generally about 8 ft. wide, and carry double tracks.

Where the orebody is wide, crosscuts are run through the lode every 100 ft. In some mines where time of completion is of great importance, a special rock-drill carriage run on rails is used, on which four drills are mounted, enabling the face to be bored out quickly. The

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best of these, namely, the one used on the Central mine, is illustrated in Fig. 6. The most economical method of driving, where time permits, is to allot two faces (if available) at convenient distance apart, to each party, so as to permit the handling of the broken rock by truckers, thus limiting the miners to operations requiring more skill.

The favored system of firing is that in which the whole face is bored out before dismantling the machines, the holes for the initial cut (which is generally formed in the center of the face) being fired first, of course, and the surrounding holes in rotation, so that by varying the lengths of fuses the burden on each hole may be lightened by the preceding firing (see Fig. 7).

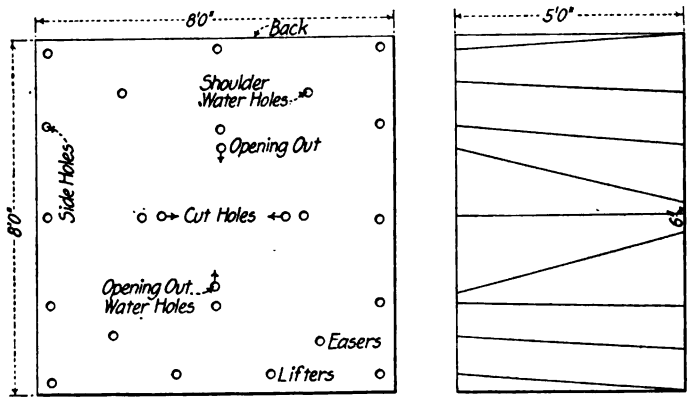

FIG. 7.—ARRANGEMENT OF HOLES FOR BLASTING OUT DRIFT.

Rises

Since the advent of the air-feed or telescopic hammer drill, vertical connections have been made to a greater extent than formerly by this means instead of by winzes. The advantages of rising are numerous. There is less risk of injury to miners, as the danger of stones or tools, etc., falling from bucket or sides of winzes is absent. The facilities for handling the broken rock are greater, shoveling on a rough bottom being eliminated. The engine-driver and braceman are dispensed with. There is improved ventilation, since the timbering of the rise can be carried closer to the face than in winzes, enabling effective use to be made of incoming fresh air whether induced by fans, air or water jets in ventilating pipes, or by curtains in adjoining drifts. The light weight of modern rising drills facilitates the work of rigging the machines in the most advantageous positions. Finally, in rises there is the absence of trouble from water; the greater facilities for detecting missed holes; and the fact that missed holes are less likely to occur in rises than in winzes, which are usually wet. Another advantage of rises as compared with

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winzes is the better ventilation of the former, heavy CO_2 gas tending to lie on the bottom of the winze.

The modern hammer drill makes much less dust than the reciprocating drill, and the small amount produced is easily laid by using water in the hole through hollow steel or water sprays playing on the collar of hole. Although for the past few years the Regulations under the Mining Act require a water supply wherever rock-boring machines are in use, it may be mentioned that for years before this rule came into force, all the hard-ore stopes in the Proprietary mine were reticulated with water under good pressure, enabling water jets to be used when boring; it was recognized that any condition tending to injure the health of the workmen called for remedial measures, not only for humanitarian reasons but in the interests of the industry generally.

It has sometimes been sought to limit the height of rises by statute, but this is unnecessary, since excessive heights are prohibitive by reason of expense in handling tools, timber, etc. Various systems of timbering are employed. In some cases the rise is simply divided into two compartments by timber, lined on the side which is to be used as a chute for the broken rock. The other compartment carries the ladderway, water and compressed-air pipes, and 10-in. pipe for ventilation. Two square sets are also used, especially where the ground is not hard, and the extra size of the rise is therefore not objectionable on the score of expense.

Another method of timbering is the "box" rise, in which three compartments are formed in the same way as the two above described, the central compartment being used as a chute, and kept always nearly full.

By placing a curtain in the drift or crosscut opposite the chute, the air is made to travel up one side of the rise to the top and down the other. The extra length of this class of rise leads to increased cost, but those favoring it claim an advantage through having no ventilating air pipes to trouble about and to protect when firing—a precaution often neglected by the men.

Trucks

A simple form of ore truck is used where horse traction is employed. The bearings are so made as to permit the automatic greasing of the axles in passing revolving greasers. End- and side-tipping trucks are used for handling mullock, etc., and for filling stopes.

Methods of Stoping

Where the ore is friable, and therefore not self-supporting, the square-set system is now universally adopted, using sets of sawn 10 by 10-in.

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Oregon pine, generally 6 ft. square and 8 ft. high from center to center. Because of several serious underground fires, attempts have been made, especially on theProprietary mine, to replace this system by the crosscut and others. In the former case, successive horizontal slices about 6 ft. wide by 7 ft. high were taken out and filled, and any drift timber used in the lower floor was drawn when the floor above was being stoped, tapered legs being used to facilitate this. Owing to the increased labor involved to both miners and mullockers, and the slow rate at which large bodies of ore can be worked, square sets were reverted to.

In places where the ore was sufficiently self-supporting, sloping stopes taking out slices 10 to 15 ft. wide were tried. The cost of handling the ore and mullock filling was very low; but by reason of variations in the strength of the ground what appeared a perfectly safe width at one time might be unsafe later on; and this system was also discarded, except in isolated cases.

In using square sets for stoping these large masses of ore, it was the rule in the earlier days to work out the ore in stopes 100 ft. high. Although this has been successfully done in many cases, it is now held by those who have had the heaviest ground to work that in the long run it is much safer and more economical to divide the height into two lifts of 50 ft. each. When stopes as high as 100 ft. are worked, the accumulated shrinkage due to the drag of the filling on the timbering in the stoped-out ground is very apt to leave considerable cavities between the old bottoms overhead and the top of the stope, rendering the blocking at the top of the stope useless. In some cases the old bottoms and filling above gradually subside and harmlessly follow the top of the stope below, but in too many cases there is a sudden drop of hundreds of tons on the top of the working stope, the effect of which may be its complete collapse, and great expense in making fresh arrangements to take out the ore, much of which will have fallen into and mixed with the filling, besides being much more expensive to mine than if the collapse had not taken place.

Where the ore is sufficiently cohesive to enable driving laths to be dispensed with to support the back, overhead stoping is practiced, because the ground can be blasted to greater advantage, and with less explosives per ton, than in underhand stoping. In heavy ground, however, or, as a rule, where a slice of ore under old stope bottoms is being taken out, underhand stoping is carried out, the miners first securing the ground immediately above the set or sets to be carried down, this securing being done with the aid of cantilever booms supported on posts set on the cap at the bottom of the set, driving laths being used if the condition of the bottom requires them (see Fig. 8).

The back or backs of the sets, if more than one are being carried down at a time, having been secured, the ore is then removed for the

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full dimension required to accommodate the square sets, horizontal breasters of 10 by 2-in. timber with joggled ends being inserted to secure the sides; and when these reach the floor of the set, the square sets themselves are placed in position and blocked. Before taking out the next lower set, the bottom timbers of the square set, in the case of the Proprietary and some other mines, are hung from the top timbers by nailing two laths to each pair of timbers, or, as in the Central mine, by connecting adjoining timbers by means of iron rods about 1 ft. long having spiked ends, about 4 in. long, bent at right angles, that can be driven into the timber. The broken ore is put into chutes formed in the square sets in the usual way, and moved as the stoping advances.

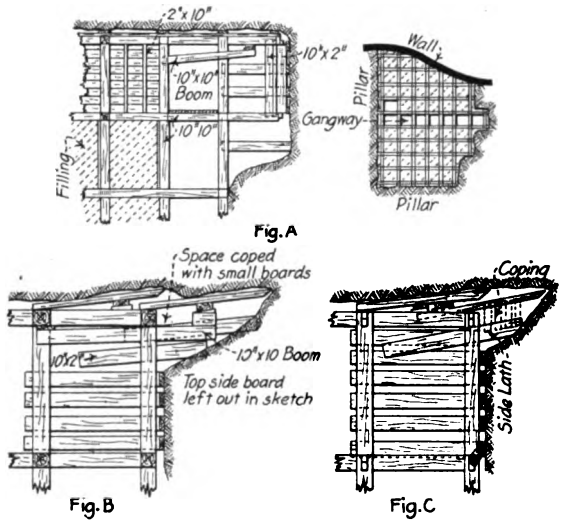

FIG. 8.—(A) METHOD OF STRENGTHENING PADDOCKING LATHS AND DETAILS OF UNDERHAND STOPING. (B) DETAILS OF DRIVING LATHS, BOOM ETC., SUPPORTING BACK IN RUNNING GROUND. (C) DETAILS OF DRIVING LATHS, BOOM ETC., SUPPORTING BACK IN RUNNING GROUND.

When all square sets from top to bottom of the run have been put in, the hanging laths and irons are, of course, removed and used in other sets as required. There is very little strain on these hangers as all the top weight is taken by the boom in the top set and the weight of the sets below is taken up by the blocking.

In stoping ore of a dry sandy nature, common in the oxidized zone of the Proprietary mine, great assistance is obtained from the use of water, sufficient of which is used in a narrow trench to soak into and wet the ground against which the breasters will be placed. By this means the danger of a run of ground is reduced to a minimum.

Where back-laths have to be driven in dry running ground, the work is

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greatly facilitated, besides being made healthier, by wetting the material in the back with the aid of pointed pipes having outlets on the sides.

Another practice in vogue in the Proprietary and other mines is the use of screw jacks for driving the laths "home." Hammering not only destroys the lath before it gets home, if driving is difficult, but the vibration tends to set up a run of ground.

With regard to the location and size of square-set stopes, the practice generally adopted is to carry the stope across the full width of the orebody, so that the stoping advances in the direction of the course of the lode, thus minimizing the effects of pressure on the stoping from the hanging wall, which, of course, becomes heavier as stoping proceeds. The extent to which stoping is allowed to advance before being filled depends on the prevailing conditions; but, in general, two sets wide by as many sets across the stope as possible are filled at each mullocking, leaving one vacant run of sets against the face. In the case of heavier ground, it is often necessary to paddock off the sets close to the working face, leaving single vertical gangways for the full height of face to give the required points of access to the face for re-starting the stoping.

The material most commonly used for filling is the residues from the flotation plants, but in the case of the Central mine, where there is much heavy moving ground and where mullock filling is obtained cheaply from a quarry immediately over the workings, the use of tailings was discontinued. Where tailings are used, as in the Proprietary mine, the sets are closely lined with 10 by 1-in. Oregon planks, and these are strengthened by two upright 10 by 2-in. laths wedged tightly between the caps. Where mullock filling is employed, 5 by 1.5 to 2-in. laths spaced 4 in. apart are used, buttressed where necessary in the same manner as is done with the 1-in. planking.

These buttress pieces are removed just prior to the filling of the succeeding stope. The bottoms of the square sets are covered with 2-in. planking, the space between which and the solid ore is filled with ore, to prevent breakage of the planking under the pressure of the overlying filling.

Open Stoping with Bulkheads

Where the sulphide ore is still in its original solid condition, free from decomposed veins and vugs, stoping is carried out without the use of square sets except for chutes and ladder-ways. The stoping begins at the various crosscuts, and where these are not already connected by driving along the orebody, a stope about 20 ft. wide by 16 ft. high is carried each way, generally along the foot wall, with a view to connecting with the adjoining crosscut and establishing ventilation. The sill-floor stope is gradually extended until the entire width of the lode is stoped out for the height above mentioned. These excavations frequently

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exceed 100 ft. in width, and may in some cases attain a length of 300 ft. Pigsties, or bulkheads, 5 or 6 ft. square, of 10 by 10-in. timbers crossed, are built from 12 to 15 ft. apart to support any ore likely to flake off. Excavations of this size are not common, as it generally happens that pillars of ground, either poor ore or country rock, occur that serve to support the back. No hard and fast rule can be laid down as to the size of these excavations, so much depending on the nature of the ore, its freedom from floors or horizontal cracks or seams, the nature and configuration of the inclosing country rock and walls, which may converge as depth increases, or *vice versa*, making wide stopes more or less safe respectively.

The extent to which pillars are left, whether because too poor to mine or because needed temporarily as a support to a shaft or other working, is also a factor affecting the area which can safely be undercut. Each case must therefore be decided by the management in view of all the conditions.

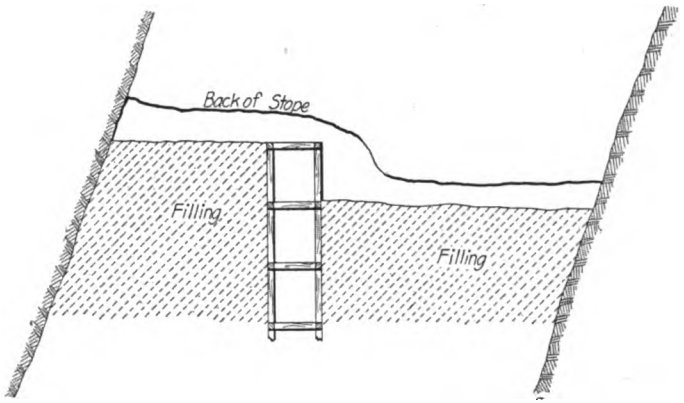

FIG. 9.—OPEN STOPE IN TWO FLOORS WHERE HANGING WALL IS WEAK.

The work of placing in position the permanent main drift and crosscut timbers can proceed as soon as blasting operations on the side of the stope are sufficiently remote to permit this to be done without risk of the timbers being thrown down, and when these have been placed and a sufficient area of filling has been put into the stope, the second stope can be started, allowing the rest of the sill-floor stoping and filling to proceed simultaneously with the stoping of the second slice. This and each subsequent horizontal slice is taken out 8 ft. high. The general practice of the field is to cover the sill floor of the stope, *i.e.*, on the level, with 4-in. planking to enable the stoping below to approach the old bottoms with safety; possibly 3-in. planking might be substituted to advantage.

The filling is taken out of chutes into end- and side-tipping trucks, and tipped over the whole area of the stope, and the space between the top of filling, *i.e.*, the rail level and the back of stope, usually from 4 to 5 ft.

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high, is again dotted with pigsties to support the back. The bulkheads are usually removed as the filling advances, although occasionally one is left and buried should the condition of the back render this necessary. It is considered safer to build bulks at regular intervals rather than only at such places as the supervising officers consider necessary, but to increase the number if occasion requires it.

In stoping hard ore against a weak and overhanging wall in the Proprietary mine, the open stope was worked in two floors (Fig. 9), the higher of which was against the wall in question, and by this means a minimum extent of wall was left unsupported, as filling was introduced to support the wall as early as possible, and the back, as seen in cross-section, was thus kept in the most favorable shape to resist and support the hanging wall.

The height to which these large stopes can safely retain their full

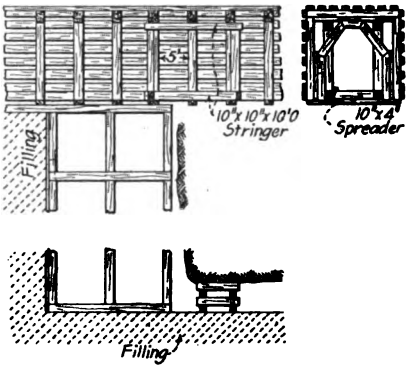

FIG. 10.—METHOD OF SUPPORTING DRIFT TIMBERS WHEN BEING UNDERMINED BY STOPING.

width depends on the conditions regulating their extent in the first instance, but the local practice is to discontinue using the open stope or bulkhead system when there remains from 25 to 30 ft. of ore under the level above. This slice is stoped out on the square-set system, underhand, if the ore is sufficiently cracked and broken to require it (Fig. 10). The bottom of the square sets is laid on the filling, and especially where tailings are used, scrap timber is first laid to give added support.

As the mining of these arches involves less heavy firing than in the open stopes, light reciprocating drills of various types, weighing about 120 lb. and capable of being quickly rigged on stretcher bar, are generally used. The ore in these cases is more or less broken as the result of the enormous pressure concentrated on the arch from the hanging wall, and in consequence does not lend itself to heavy firing, especially in contact with the square-set timbering which might be shot out, bringing about the loss of the stope. In some cases, the ore is sufficiently cracked

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and broken to make the use of hammer drills held in the hand advantageous. Difficulty is met with in getting the miners always to use water with these drills in the interests of their own health, and this also applies to the ordinary "popping" of rocks for which these drills are also used.

For the short holes necessary, the drills will generally clear themselves with the aid of the water spraying into the hole, and the exhaust air from the machine; but in many cases rather than submit to being slightly splashed with mud, men risk their health by working the machines dry and inhaling dust unnecessarily. It is too often forgotten that when hand drilling was employed for popping purposes the mud had to be periodically scraped out, and in those rare cases in which the popper does not clear itself the hole can be scraped out as of old.

Reciprocating rock drills with cylinders 3¼ in. in diameter are generally used in the solid ore, since with these, approximately horizontal holes can most conveniently be bored in the backs, enabling regular and reliable roof to be formed. Telescopic drills were tried for general stoping, but these tend to leave an unsafe back.

As many holes as practicable are bored at each setting up, and these are fired by ordinary fuses cut to lengths to give the desired rotation of shots, blasting gelatine (92 per cent. nitroglycerine) being used so as to effect the maximum of shattering of the ore to save spalling. For the same reason, simultaneous firing with electric fuses was, after making tests, found uneconomical, as the ore came down in much larger pieces entailing added expense in "popping" and spalling. The firing of one hole at a time results in numerous cross-breaks in the ore and reduces the cost of breaking up the ore to the size required by the mills, *i.e.*, about 10 in. in diameter.

Another disadvantage of electric firing in stopes was that the excessive vibration set up in firing all the explosives at one time was a source of danger, as it would tend to weaken the backs of stopes and lead to accidents through falls of ground.

A modification of the open-stope or bulkhead system of stoping was made use of on the Proprietary mine, where it was desired to obtain the maximum output of ore from a given length of lode; this applied particularly in exceptionally hard portions of the lode, where the cost of sinking winzes was excessive, and the lode had less than the usual width. The system may be described as an adaptation to lode mining of the long-wall principle in coal mining. Several floors of stoping are worked at the same time, access to each face from the main mullock chute and ladderway being preserved by the provision of timbered drifts connecting the mullock chute with each stope. In this manner, any number of faces can be worked from each main mullock chute, each on a separate floor, but the maximum number actually operated was four.

The system was not extended, since the introduction of the Calyx

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drill enabled mullock passes suitable for sand filling to be provided at about one-quarter the cost of winzes. The holes bored by the Calyx drill were about 10 in. in diameter.

Rill Stopes

A certain amount of rill stoping has been practiced on the field where conditions favored that system of work, but this is not usual. The hanging wall of the lode is oftener weak than strong, and constitutes a menace to the workmen employed on the sloping surface of the filling, especially when near the bottom of the rill where the danger is greatest, owing to the length of time the wall has been exposed.

The nature of the filling generally available, namely, sandy tailings which never form a compact surface, would necessitate the covering of the stope with planking to prevent undue mixing of broken ore with the filling.

Further disadvantages as compared with flat back-stopes are the greater difficulty of rigging the rock drills, and less favorable conditions for blasting, in that the ore is inclined as a rule to lie in horizontal layers, and lastly, greater difficulty in breaking and keeping separate worthless material occurring in the lode.

Transportation of Filling

The handling of mullock filling from the main chutes to the various stopes is to a great extent accomplished by horse traction on the Proprietary and some other mines, but where the distances are small hand trucking is more convenient.

Conveyor belts are used in a number of cases, particularly in the North mine, the Proprietary, and the Junction North, where the quantity to be handled warrants the installation. In the case of the North mine, the regularity and compactness of the orebodies has enabled the management to install a complete system of conveyors for delivering the filling to the chutes commanding the various stopes. Where the orebodies are irregular and spread over considerable distances this system cannot, of course, be economically employed.

Electric traction was introduced on one of the principal levels of the Proprietary mine, but was replaced by horse traction after being thoroughly tested. The distances to be trucked were not sufficient to compensate for the extra cost of attendance. The horse-driver acts as a trucker, also, in getting his rakes together and requires no assistant, but the more highly paid motor-driver needs an assistant to shift points, couple and uncouple trucks, etc. In addition, repairs, when necessary, are costly.

Having arranged the gradients of the levels so that a horse can draw the same number of loaded and empty trucks with equal ease, *i.e.*,

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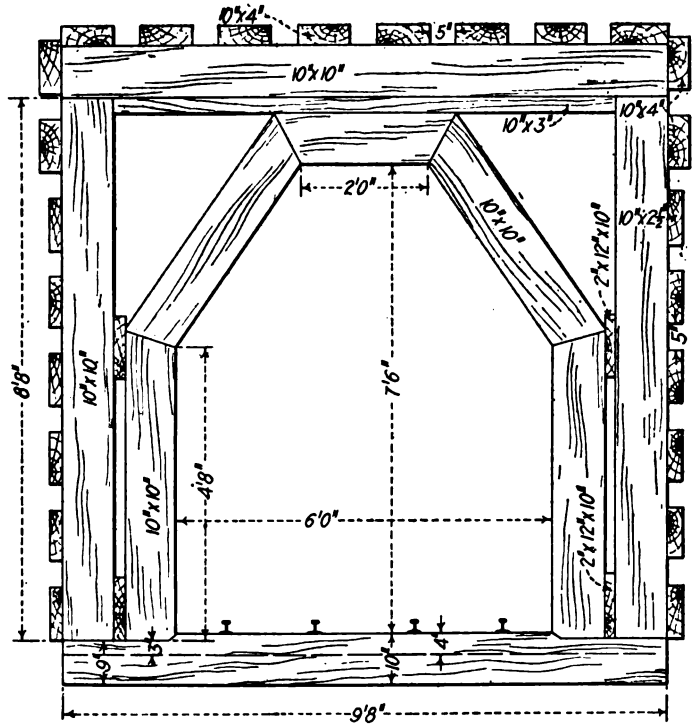

FIG. 11.—STANDARD DRIFT TIMBERING FOR DOUBLE TRACK, BROKEN HILL PROPRIETARY CO., LTD.

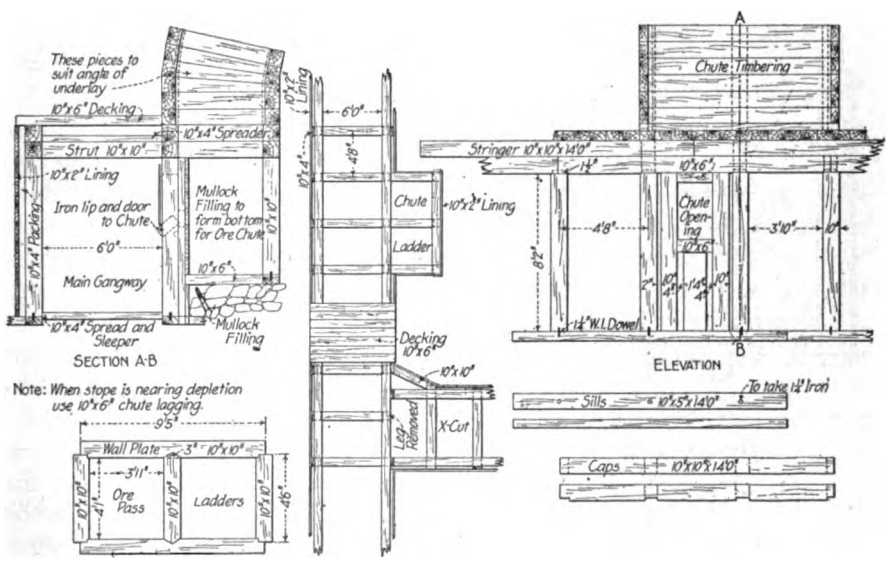

FIG. 12.—UNDERGROUND LEVEL TIMBERING, THE BRITISH BROKEN HILL PROPRIETARY CO., LTD.

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about 1 in 190, it is found that one horse can thus handle 10 trucks at a time containing 25 cwt. each.

Main-Drive Timbering in the Orebody

Various types of timbering for the main drives are employed, all of which give satisfactory results. Where hardwood timber is obtainable at

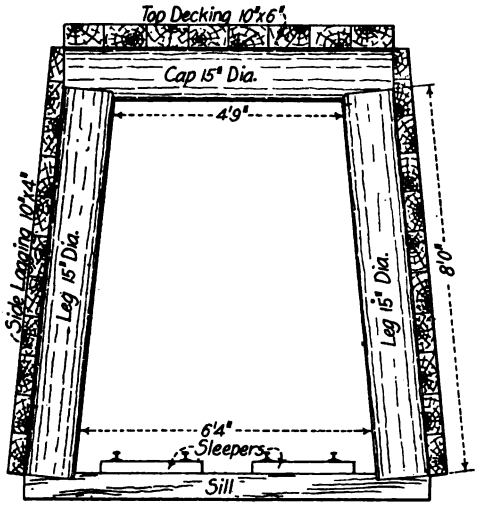

FIG. 13.—STANDARD DRIFT TIMBERS, SOUTH MINE.

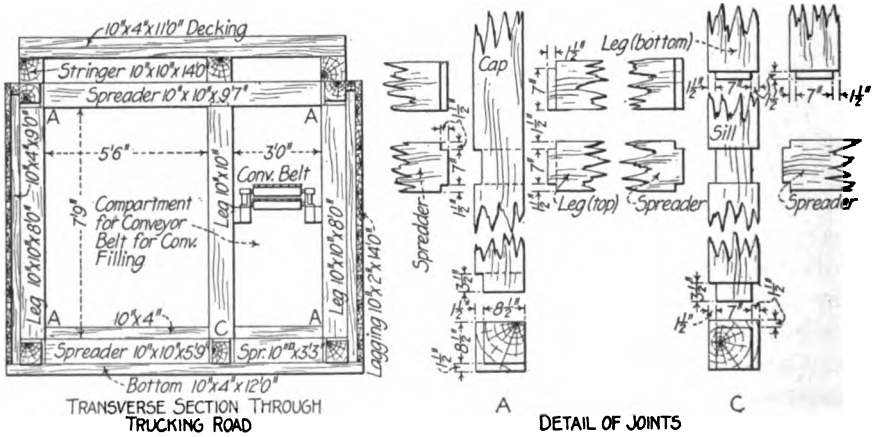

FIG. 14.—MAIN LEVEL TIMBERING, NORTH BROKEN HILL, LTD.

reasonable rates compared with Oregon, its use would be advantageous in these places. The principal methods of timbering main gangways in the lode are illustrated in Figs. 11, 12, 13, and 14.

A number of the mines use a considerable quantity of round hardwood

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logs for ordinary drives in heavy ground, as their resistance to crushing is much greater than that of Oregon. On the Proprietary mine these are used in conjunction with old 80-lb. rail material as caps; this is specially advantageous where head-room is deficient.

Timbering of Ore Chutes

The majority of the ore chutes in the stopes, whether square-set or open stopes, are formed by two square sets lined with 10 by 4-in. hardwood planks, spiked to the caps in a vertical position. The cost of repairs is reduced materially by the use of hardwood, in spite of its extra price.

The use of square sets is advantageous in that repairs to the lining of chutes can be readily effected. On the British mine these chutes are built of 10 by 10-in. timber, having joggled ends, the idea being that the

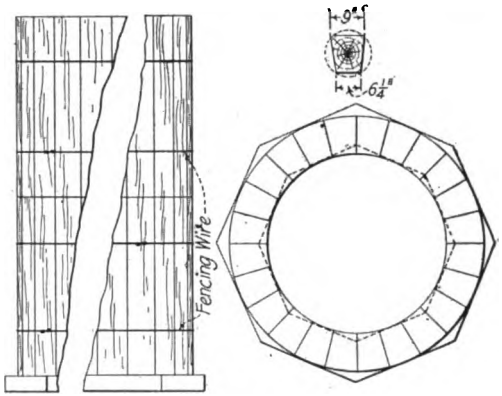

FIG. 15.—CIRCULAR ORE PASSES, SOUTH MINE.

timber will last out the stope. Against this, there is greater wear on the timber when the grain is at right angles to the falling ore.

With square sets a convenient construction is afforded for the provision of a ladderway alongside the chute, which, besides being useful for traveling, is of great assistance in case the ore in the chute "hitches up." The advocates of chutes without ladderways claim that the material can generally be started with the aid of a cannon, but this involves a certain amount of delay, especially as the cannon is not always effective.

On the South mine, where the ore occurs to a greater extent in pipes or chutes, circular ore chutes built up in sections, as shown in Fig. 15, are successfully used. The timbering is thickest, of course, near the bottom, where the wear is greatest. An essential condition for the satisfactory use of these circular chutes is that the pressure be approximately equal on all sides; their use, therefore, in ground subject to much pressure from the hanging wall, is prohibited.

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Prevention of Underground Fires

The leading mines, where considerable quantities of timber are used underground, have complete installations of fire-service mains, generally 1¼ in., throughout the various workings, with tanks underground, placed, say, every 300 ft., to avoid excessive pressure on the mains, and kept full by means of ball float-valves.

The local experience with regard to underground fires has been that the only hope of subduing them is by the prompt application of a small quantity of water under pressure, and therefore the lighter the gear the more likely it is that the fire will be subdued before it reaches portions of the workings that might quickly become inaccessible through the burning out of the timbering. For this reason, ¾-in. hose in 60-ft. lengths, placed in the levels every 200 ft., is used on the Proprietary mine.

With a view to preventing the spread of fires which may obtain too

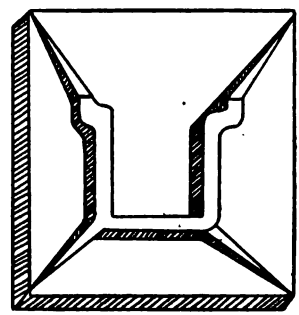

FIG. 15.—CASTING AT THE JUNCTION OF SET TIMBERS TO HOLD THE END OF A SPREADER.

strong a hold on square-set stopes to permit extinction without flooding the workings (which is sometimes impracticable), the Proprietary Company has made a practice of forming, at various portions of the workings, barriers to the progress of fires by having all timber connecting adjoining sets replaced with iron or steel spreaders, generally railway rails; and special plain castings (Fig. 15) are placed at the junction of the set timbers to hold the end of rail in position and prevent them entering the timber by increasing the area of base. If these barriers are constructed while ordinary stoping proceeds, very little added expense is incurred, as there is generally a stock of old rails available at low cost.

The various mines have provided themselves with different forms of smoke-jackets to enable the wearers to enter workings containing an irrespirable atmosphere, goggles being also provided to protect the eyes from smoke. Among those employed are the "Proto" and "Pneumatogen" types, using compressed oxygen, and the "Aerophor" using

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liquid air; the last named has only recently been introduced at the North mine.

This apparatus is looked after by the fire station and ambulance staff, resident at each large mine, under the supervision of the underground manager, and in addition to the members of these staffs the shift bosses receive periodical practice with the breathing apparatus. Each of these mines possesses pulmotors, in the use of which the ambulance staff and others are trained, so that respiration can be most effectively restored in cases of "gassing," etc.

Underground Staffs

The problems to be faced in the working out of the large bodies of ore occurring in this district call for the application of scientific principles to a much greater extent than is the case with narrow lodes.

In the earlier days when the exploitation of the orebodies was not systematized to the extent that now obtains, creeps due to the collapse of stopes were frequent. On this account it has been the practice for a considerable number of years to put in charge of the underground work an officer who has received thorough scientific training as a mining engineer, combined with sufficient practical experience. In some cases this consisted of 18 months' manual work in various capacities underground, etc., in other cases, three or four times this length of service as underground surveyor. This officer usually has a similarly trained assistant, or an understudy is being trained while on the underground survey staff.

Under the underground manager and his assistant, there are day foremen, as many as necessary, according to the extent of the workings allotted to him. To these men fall the duties of letting all the ordinary ore-breaking and development-work contracts, besides looking after the general safety of the workings, arranging for the filling of stopes as required, and many other duties that it is not necessary to particularize.

Next, there are in the largest mines assistant foremen who are competent to take the place of the day foreman in case of his absence; and these men follow each shift, the one on the day shift generally having charge of the wages-timbermen who, with few exceptions, work on day shift only. Finally, there are the shift bosses who are picked men drawn from the ranks of the miners and who may ultimately become foremen. In larger mines there is a separate boss in charge of the truckers, both on ore and in filling stopes—the ordinary boss confining his attention to miners' work.

An important part of the duty of shift bosses is to see to the safety of the backs, etc., of the stopes. Familiarity with danger too often leads to contempt of it on the part of the miners, and consequently to accidents.

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Timekeepers check the attendance of the workmen when entering and coming up the shaft, and make up distribution sheets showing the branch of work against which their earnings are to be charged, *i.e.*, stoping, development or preparatory work, repairs of different kinds, etc., as the case may be.

The contracts are let as a rule to parties of six men, *i.e.*, two per shift; but sometimes this number is doubled especially if separate chutes are not available. As a general rule, a small party works more amicably than a large one, and for this reason is preferable. The earnings are made up and distributed to the various individual members of the contracting party by the companies *pro rata*, according to the number of shifts worked, thus relieving the party of the necessity of accurately apportioning the proceeds of the contract.

Costs

Complete cost sheets are made up showing the cost per ton of the various items of expenditure, there being subdivisions into: (1) Labor; (2) Stores (including coal and water for power and all supplies); (3) Miscellaneous, and (4) Special expenses not directly chargeable to ore raising, but which must be included in the total cost.

At the Proprietary mine, complete costs are worked out and made available to the management two days after the close of the week; and, in addition to the complete weekly costs, daily cost sheets on somewhat less elaborate lines are also made up; most of the items in these daily costs are accurately stated, but in a few instances where it is difficult to obtain the daily quota, average figures are used.

The total cost of mining in the district varies considerably according to the nature of the ore and its mode of occurrence—whether in large regular masses or separate veins, or in pillars left between old stopes, or in isolated bodies. But, in general, it may be said that the costs range from 15 to 20 shillings per ton when, as during 1914, the earnings of miners on contract averaged about 17s. 6d. per 8-hr. shift.

In conclusion, I wish to acknowledge the courtesy shown by the managers of the various other mines in facilitating the preparation of this paper.

Addendum

The practice at the Broken Hill Proprietary mine with regard to rock-drill hoses has recently been altered. Formerly these hoses were surrounded with a protecting cover of marline or woven cotton. During the last two years these hoses (which are 1 in. internal diameter, having 6-ply canvas) have been made with a covering of ⅛ in. thickness of rubber on the outside, of a quality similar to that used for motor car

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tires and high-quality conveyor belts. It was thought that inasmuch as rubber has proved to be the best material for resisting abrasion on motor car tires, it should also give the best service for rock-drill hoses, which are subjected to wear through being dragged over rough ground. The slightly higher cost of these hoses has been amply justified by the results obtained. These hoses are practically indestructible so far as exposure to mineral water is concerned, whereas the old type of protection was quickly destroyed by the influence of mineral water.

Discussion

Albert R. Ledoux, New York, N. Y.—As a contribution out of my ignorance, I would like to ask whether it is customary to illuminate shafts in this country by reflected light from above through a mirror as described on p. 55 of this paper. I have used that system in mine surveying where the shaft was very wet, and it was difficult to employ candles. I placed a mirror at the top of the shaft, at an angle of 45°, and not only by this means illuminated the bottom, but by placing another mirror in the shaft at the bottom, I was able to throw light into the adit tunnel, and I wonder if that practice is common in this country, either in surveying or shaft sinking?

W. L. Saunders, New York, N. Y.—As far as my experience goes, it is not at all common; it is very rare.

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Underground Mining Methods of Utah Copper Co.*

BY THOMAS S. CARNAHAN, B. S.,† BINGHAM CANYON, UTAH

(New York Meeting, February, 1916)

THE mining property of the Utah Copper Co. is situated in the West Mountain mining district, Salt Lake County, Utah, in the Oquirrh Range of mountains.

GEOLOGY

In a general way the rock formation of the district consists of a series of beds of quartzite and limestone intruded by a body of monzonite porphyry roughly elliptical in shape, with an east-west axis over a mile in length and a north-south axis of about 3,000 ft.

This porphyry intrusion, accompanied by strong mineralizing action and fracturing, resulted in the formation of orebodies in the adjacent sedimentary rocks, and was itself sufficiently mineralized to make it one vast orebody. The Utah Copper Co.'s mining property comprises within its boundaries practically the entire outcrop of the monzonite mass, of a commercial grade.

GENESIS OF PORPHYRY ORE

Following an intricate system of fracturing, mineral solutions circulated freely through the porphyry, depositing small quantities of copper and iron, and resulting in a considerable silicification of the monzonite. The quantity of copper originally deposited was undoubtedly too small to have ever given the porphyry a commercial value, had not secondary enrichment, due to the leaching of the copper from the surface of the mass and re-deposition in the sulphide zone, been active over a long period of years.

A large portion of the leached material has probably been removed by erosion, but there still remains a blanket of leached and oxidized porphyry of varying thickness covering the sulphide ore, known as capping. Over certain sections of the orebody, this capping contains commercial quantities of copper carbonates, but most of it contains little or no copper.

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* Originally presented at the annual meeting of the Utah Section, Salt Lake City, Oct. 4, 1915.

† Mine Engineer, Utah Copper Co.

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To Jan. 1, 1915, a total of 377,690,000 tons of ore had been developed, of which 342,500,000 tons averaging 1.45 per cent. copper still remained to be mined. The average thickness of the developed ore was 465 ft., while the layer of capping covering the ore averaged 115 ft. Further development will undoubtedly show an increase in the average thickness of the ore, with a corresponding increase in the tonnage of developed ore.

Underground Mining Auxiliary to Steam-Shovel Operations

As is generally known, the Utah Copper mine is primarily a steam-shovel operation, and it will perhaps surprise many that up to April, 1914, a considerable tonnage of ore was obtained by underground mining methods.

During the early years of steam-shovel mining the amount of ore available was naturally limited, since most of the shovels were working in capping, and it was necessary to stope a large tonnage underground in order to keep the mills at Garfield running at capacity.

During the 3-year period from 1911 to 1913 inclusive, a total of 102,719 ft. of drifts, raises, etc., was driven on the property. Most of this development served the double purpose of proving the shape and value of the orebody, and providing the necessary openings for stoping operations. The output of ore from underground operations in these 3 years amounted to 3,071,719 dry tons, of which 247,280 tons came from development and the rest from stopes.

Shrinkage Stoping System Adopted

Realizing that underground mining was to be but an incident in the mining of the orebody as a whole, a system of stoping was adopted which would not affect adversely future steam-shovel operations. In order to fulfill this requirement it was essential that the surface should not be caved, that no large openings be left unfilled, and that the capping should not be mixed with the ore.

The system as finally adopted and successfully operated, consisted in starting stopes on three separate levels or tunnels. The first of these tunnels, at an elevation of 6,733 ft. driven 7 by 7.5 ft. in the clear, was the main or motor-haulage level. The second, at an elevation of 6,883 ft. or 150 ft. vertically above the main level, was equipped for hand tramming only, so all drifts, crosscuts, etc., were driven 5.5 by 6.5 ft. in the clear. The third, at an elevation of 6,983 ft., or 100 ft. vertically above the second, was also a hand-tramming level and driven 5.5 by 6.5 ft. in the clear. These three levels were connected by many manways and raises for dropping the ore from the upper tunnels to the motor-haulage level. An underground shaft centrally located, equipped with a com-

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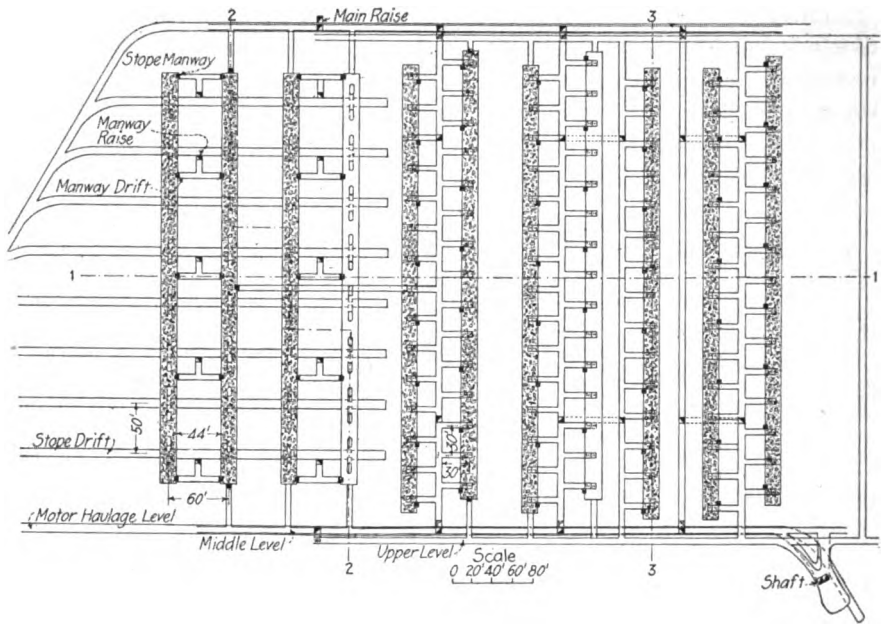

FIG. 1.—PLAN OF WORKINGS FOR ONE BLOCK OF STOPES.

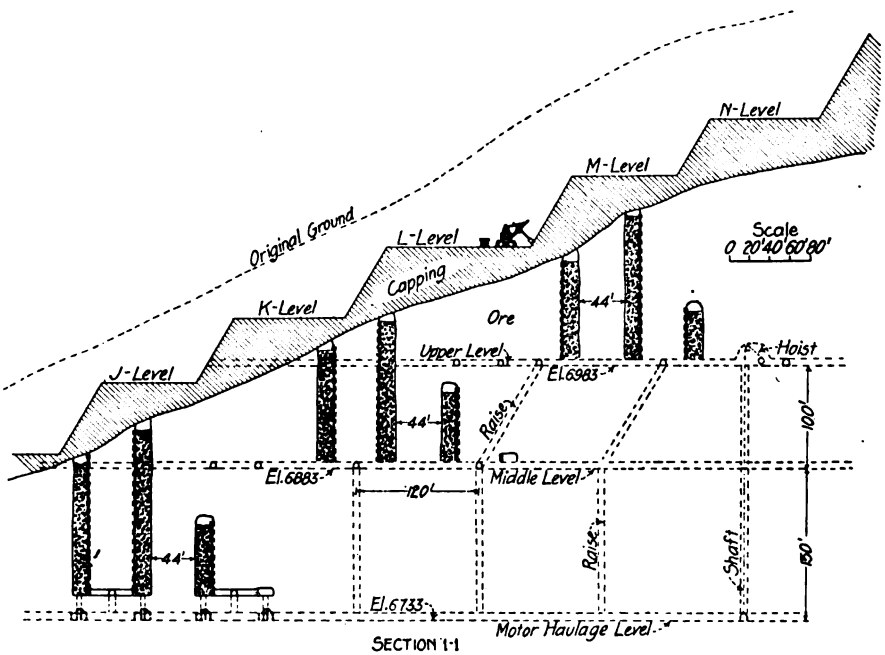

FIG. 2.—SECTION SHOWING RELATIVE POSITION OF STOPES TO STEAM-SHOVEL LEVELS.

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pressed-air hoist, was used to hoist supplies from the motor-haulage level to the upper levels, but no ore was handled through it. Each level had one or more surface connections, affording good natural ventilation to all parts of the workings (Figs. 1 and 2).

Orebody Worked on Three Levels

By means of these three levels, the orebody was divided into blocks for stoping, the central block, 500 ft. wide (see Fig. 1), was bounded on the main level by two parallel motor drifts, and on the upper levels by two main parallel tramming drifts, directly over those on the haulage level. At intervals of 120 ft. along these motor drifts, raises 5 by 6 ft. were put up on a 55° pitch to the level above and afterward extended to the third level.

In order to make the stopes as safe as possible, to minimize the amount of timber required, and to leave substantial walls for the safety of future steam-shovel operations, it was decided that the standard width of stope should be 16 ft., with 44-ft. pillars between stopes.

The method of starting stopes on the motor-haulage level was somewhat different than on the upper levels although the size of stopes and pillars was the same.

Main or Motor-Haulage Level

Motor drifts were driven on the motor-haulage level, spaced at 50-ft. centers, and parallel to the main drifts forming the boundaries of the block. At intervals of 60 ft. along these drifts the surveyor marked the center of the stopes as the drifts were driven; if the ground required timbering the tunnel sets were spaced to be suitable for stope-chute sets later on. After the stope-chute sets were completed, a man with a stoping machine drilled both sides directly over the chutes, nearly horizontally and on the center line of the stope, to form a pocket at this point. The next round on each side pointed strongly upward, and from that point on, the raise was extended on a 60° pitch until the face was 31 ft. vertically above the top of rail. The stoping machine was then taken out, and a No. 9 Leyner machine set up near the top of the raise, and a drift started each way. These drifts were run horizontally following the center line of the stope, and were made 8 by 8 ft. Since their maximum length from any ore chute was only 18 ft., little shoveling was necessary to get the ore to the chute. After this drift was completed for the full length of the stope, the Leyner machine took 4 ft. off each side of the drift, to bring the stope to the standard width of 16 ft. The ore broken in all this work was drawn from the chutes, so that when this stage of operation was completed, an excavation 450 ft. long, 16 ft. wide and 8 ft. high was ready for stoping.

Chute timbers were constructed as follows: Three tunnel sets of 12 by 12-in. timbers were set up, spaced at 5½-ft. centers. Posts 8 ft. long

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were set in hitches cut in the floor deep enough to make the bottoms of the caps 7 ft. above the top of rail. Caps were cut 10 ft. long, so as to extend 6 in. beyond the side of each post, and blocked tightly against the walls. Planks, 2 by 12 in. by 7 ft. long, nailed under the cap, acted as spreaders for the posts, but no sills were used. The sides were lagged with 2 by 12-in. planks. Enough ground was then broken above the tunnel sets to make room for a short set. The posts for this set were cut 3 ft. 9 in. long of 12 by 12-in. timber. Only two short sets were put in, one on each side of the chute mouth, the top and sides being lagged with split round poles (see Fig. 3).

Manway Raises and Drifts.—In alternating drifts, that is, at intervals of 100 ft., manway raises were put up in the pillars, midway between the stopes. The raises were started in offsets 6 ft. from the center of the track, and driven on a 50° pitch. They were made 4 by 6 ft. and divided

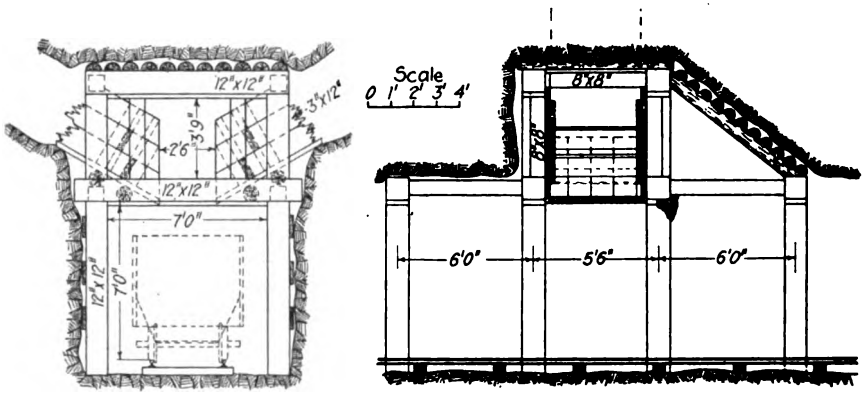

FIG. 3.—STOPE CHUTE TIMBERS ON MOTOR-HAULAGE LEVEL.

into a chute and a manway by means of stulls and 3 by 12-in. planks. The manways were equipped with ladders, and the chutes equipped with gates, so that the ore broken in the raises could be loaded direct into motor cars. When the manways reached an elevation 31 ft. vertically above the top of rail, manway drifts were started both ways at right angles to the raises, or parallel to the motor drifts. The bottoms of these drifts were 25 ft. above the top of rail, or on a level with the bottom of the stopes. These drifts, at 19 ft. from the manways, broke into the sides of the stopes. At the junction of the drifts with the stopes, 8 by 8-in. sills 8 ft. long were laid 3 ft. in the stope and 5 ft. in the drift and 6 by 6-in. cribbing built upon them, dapped 1 in. so as to leave a space of 4 in. between cribbing. The crib timbers were 4 ft. long, making the manways 3 by 3 ft. in the clear. The cribbing on the drift side was left off for the first 5 ft. to form an entrance into the manway. The manway timbers projected 2 ft. into the stope and 2 ft. into the pillar, thus placing them

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in solid ground on part of three sides. After the manways were thus completed to within a few feet of the back of the stope, and equipped with air lines, the stope was ready for active stoping.

Stoping Operations.—Usually, in working a stope, a crew consisting of a machine man and a pick man worked from each manway, so that ordinarily five machines were working in each main-level stope. Stoping was done on but one shift a day, so the same crew was responsible for conditions at a given manway. The pickman trimmed down the back of the stope, so as to make it safe for the machine man. In a shift of 8 hr., each stope crew was expected to put in from 14 to 20 holes, 6 ft. deep, depending on the ground, and load and fire them. A round of 15 holes ordinarily broke 150 tons. The motor crew on the following shift pulled about 75 tons, representing the swell of the ore, so that the stope crew

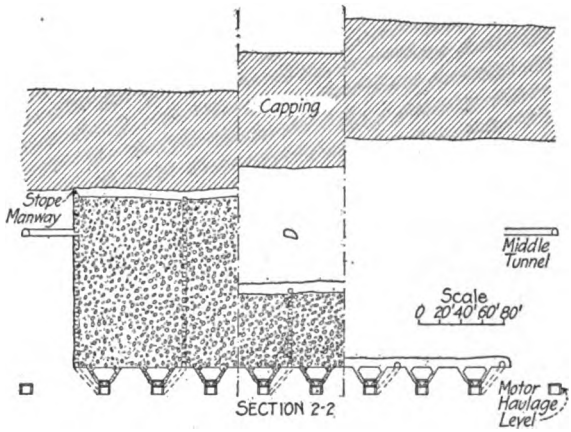

FIG. 4.—SECTION SHOWING THREE STAGES OF STOPES FROM THE MOTOR-HAULAGE LEVEL.

would have sufficient room to work on top of the broken ore. The ideal condition for efficiency and safety in a stope was to keep the top of the broken ore within 9 ft. of the back of the stope. With five machines in a stope, a separate crew of crib men was employed to build the manways, and keep them at all times well above the level of broken ore in the stope. The stopes from the main tunnel were carried to capping, or abandoned 10 ft. below the second level, if the ground above the second level had already been stoped. As soon as a stope was abandoned, the chutes were nailed up or otherwise obstructed, so that no ore could be pulled from them (Fig. 4).

Upper Levels

As already mentioned, the upper levels consist of main drifts directly above the motor drifts forming the limits of the stope blocks. Crosscuts were driven every 120 ft. at right angles to the main drifts, that is, paral-

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lel to the center line of the stopes. These were driven on a 1 per cent. grade from each main drift, meeting in the center of the block. Each crosscut was connected to the main-haulage level by two raises which came out in the floor of the crosscut 100 ft. from each of the main drifts. On both sides of these crosscuts, stope drifts were driven, at 30-ft. intervals, for a length of 30 ft., or 11 ft. beyond the center line of the stopes. They were then enlarged sufficiently to accommodate the timbers for stope chutes.

Chute Timbering.—The chute timbering consisted of three tunnel sets spaced at 4½-ft. centers, except in every other drift where an additional set was put in at 3½-ft. centers to accommodate the manway. Tunnel sets were of 10 by 10-in. timbers, with posts cut so as to leave 7 ft. between

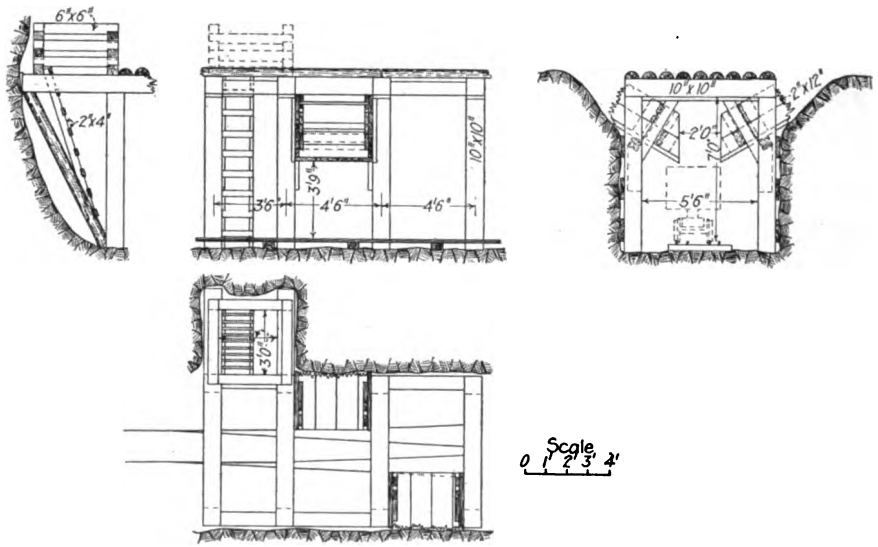

FIG. 5.—STOPE CHUTE TIMBERS ON HAND-TRAMMING LEVELS.

top of rail and bottom of cap. Collar braces were of 8 by 8-in. timbers, or round poles not less than 7 in. in diameter. Caps for manway sets were 11 ft. long to form the sill for manway cribbing. The chute lip was 3 ft. 9 in. above the top of rail and was made of 3 by 12-in. plank. Chutes were equipped with two gates so as to control the flow of ore easily. Top and sides were lagged with 2 by 12-in. plank, or split round poles (see Fig. 5).

Stope Preparation.—After the stope-chute timbers were in place, the ground above the timbers was drilled with a stope machine; but before blasting, the top was lagged with short split round poles, so that the ore could be loaded direct into a car without shoveling. An 8 by 8-ft. drift was then started on top of the chute timbers with the center line of the

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stope as one side of the drift, and extended the full length of the stope. Later this drift was widened to 16 ft.

Manways.—Manways were put up in every second stope drift built of the same dimensions and material as the manways from the main level, and carried to the third level or to capping if struck before the third level was reached. Where the distance to capping was greater than 100 ft. but less than 200 ft., connection was made with the stope on the third level, the old manways abandoned, and new ones started from the third level. Where the ore thickness was greater than 200 ft. above the second level, stopes were started from the third level, run to capping, and abandoned before the stopes from the second level were started.

Order of Working Stopes

In a given block it was the custom to have the stopes on the top level worked out considerably in advance of the stopes started from the middle

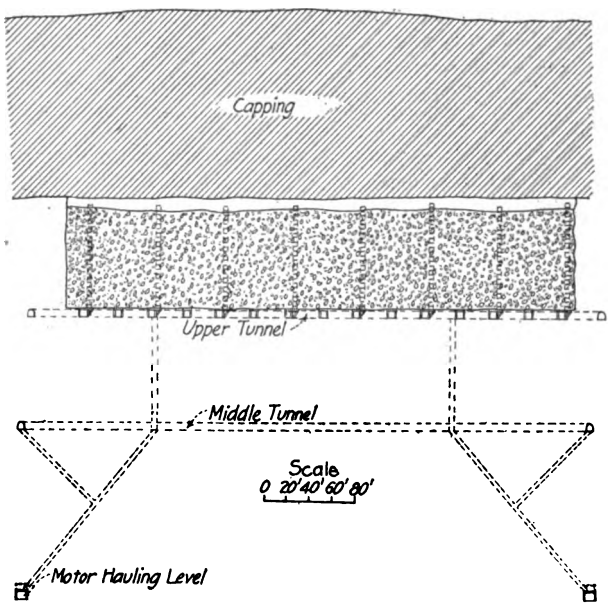

Fig. 6.—Section Showing a Stope Started from the Upper Level, and Raises Through Which the Ore is Dropped to the Motor-Haulage Level.

level. In this way the upper stopes were abandoned before the stopes from below began to disturb the tramming drifts and stope manways. This was necessary also to avoid cutting off the raises through which the ore was dumped from the upper to the main-haulage level, before the upper stopes were worked out. For the same reasons the stopes started

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from the middle level were kept well in advance of those started from the main level (Fig. 6).

Hand Tramming on Upper Levels

On the middle and upper levels all ore was hand trammed from the stope chutes to the nearest raise. The raises were so spaced that the maximum length of tram was 150 ft., and the average 90 ft. for all stope ore. Tracks for hand tramming were 18-in. gage, laid with 16-lb. rails and a grade of 1 per cent. in favor of the loaded cars. The stope cross-cuts were double-tracked for short distances near the raises, to permit several cars to run to the same raise without delaying or interfering with each other. Tram cars measured 2 by 2.5 by 4.6 ft. in the clear, and held about 1 ton of porphyry ore. Two trammers were used on a car, and the average tonnage trammed by each crew from a stope was about 65 tons per shift. A tallyman, stationed at each active stope, credited each car dumped to the proper crew.

Data from an Average Monthly Statement

Month of August, 1912.

Tonnage from stopes	112,990	
Tonnage from development	8,026	
Distribution of Labor	**Number per Day**	
Stopes:		
Machine men	41	
Pickmen and helpers	41	
Trammers	87	
Cribmen (stope manways)	11	
Timbermen (stope chutes)	11	191
Development Work:		
Machine men	28	
Muckers	32	
Timbermen	4	64
General:		
Trackmen	6	
Timbermen	4	
Motor crews	12	
Foremen and shift bosses	8	
Engineers and samplers	4	
Miscellaneous	29	63
Surface Men:		
Blacksmiths, etc	5	
Gravity tramway	20	
Common labor	6	
Miscellaneous	9	40
Total		358
Average daily tonnage per man		10.9 tons

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General Costs

During the 3-year period, 1911 to 1913 inclusive, 2,824,439 dry tons of ore were drawn from stopes at a cost of 56.6c. per ton loaded into railroad cars at the ore bin. In addition, 102,719 ft. of development work yielded 247,280 dry tons at a cost of $4.95 per foot of development or $2.05 per ton of ore. As the development was carried on entirely in average grade of ore, it about paid for itself from the ore broken. Counting every item of expense, the cost of producing a ton of ore from all sources by underground mining averaged 68.7c. By the system in use, with 16-ft. stopes and 44-ft. pillars, less than 27 per cent. of the ground developed was actually stoped, and as the stopes were full of broken ore when abandoned, the actual production represented less than 50 per cent. of the ore broken, or about 13 per cent. of the ore blocked out by the development work. The cost of production would have been considerably less, if the pillars could have been mined and the broken ore in the stopes pulled. Manway drifts and raises, and stope drifts, were charged direct to stoping, all other drifts and raises being charged to development.

Cost of Development Work

	Per Foot
Motor-haulage drifts (7 by 7.5 ft. in the clear); No. 9 water Leyner used (progress 5 ft. per round):	
Drilling and mucking	$3.50
Powder, caps and fuse	1.25
Compressed air	0.30
Miscellaneous	0.55
Total	$5.60
Timbering (where necessary)	1.61
Total	$7.21
Tramming drifts (5.5 by 6.5 ft. in the clear) 2.5-in. piston machines used (progress 3.8 ft. per round):	
Drilling and mucking	$2.25
Powder, caps and fuse	0.68
Compressed air	0.21
Miscellaneous	0.47
Total	$3.61
Timbering (where necessary)	1.35
Total	$4.96
Raises (4 by 5 ft.); hammer machines used, (progress 5 ft. per round):	
Drilling and tramming	$1.50
Powder, caps and fuse	0.85
Compressed air	0.15
Miscellaneous	0.90
Total	$3.40

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All development work was contracted at the following rates for breaking and mucking, not including timbering, the company furnishing the supplies:

Motor drifts (7 by 7.5 ft. in clear)
$3.50 to $4 per foot (depending on whether dry or wet ground)

Tramming drifts (5.5 by 6.5 ft. in clear)
$2.25 to $2.50 per foot (depending on length of tram)

Raises (4 by 5 ft. in clear)
$1.20 to $1.80 per foot (depending on length of raise)

At these rates, the contractors were able to make from $3.75 to $5 per shift, depending on the ground and their own efforts.

General Distribution of Costs

	Per Dry Ton
Labor	$0.329
Powder, caps and fuse	0.091
Timber	0.049
Motor haulage	0.025
Gravity tramway	0.020
Supervision and engineering	0.038
Compressed air	0.024
Miscellaneous	0.111
Total	$0.687

These figures include every item of expense incident to the mining of the ore, including the necessary development work. Under the head of miscellaneous is included the proper proportion of all general expenses, such as taxes, insurance, Salt Lake and New York office expenses.

Hammer and Piston Drills Used

For all large drifts, such as motor drifts and stope drifts, and for widening out stopes, No. 7 or No. 9 Leyner water drills were used, the smaller machines being used only in soft ground. For small drifts the one-man 2.5-in. piston machine, of both Ingersoll and Sullivan makes, was used, while hammer drills were used for raising and stoping.

Water Supply

In the principal drifts and crosscuts on each level a 1½-in. pipe line was laid to supply water for the Leyner machines and for fire protection. A reserve supply of water was stored in tanks erected in a drift 50 ft. above the top level, the head being sufficient to carry the water to all parts of the workings. As the two upper tunnels were dry, it was necessary to fill the storage tanks by pumping from the ditch on the main-haulage level, a small compressed-air pump operated an hour or two each day being used for the purpose.

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Data on Motor Haulage

The main-level track is laid with 60-lb. rails, 36-in. gage, and a ¼ per cent. grade in favor of the loaded cars. Two General Electric 10-ton locomotives, each pulling a train of seven cars or 60 tons of ore per trip, have handled as high as 4,000 tons in a shift of 8 hr., the average being 2,500 tons. Loading from a chute in which the ore runs freely, a train could make a round trip in 15 min., divided as follows:

	Minutes
Loading train at ore chute	2.5
Trip to ore bin (4,000 ft.)	5.0
Dumping train at ore bin	2.5
Return trip to chute	5.0
Total	15.0

Gravity Tram from Mine Portal to Railroad Bins

As the portal of the motor-haulage tunnel is on a hill side, at a vertical distance of 350 ft. above the level of the railroad yard, and at a horizontal distance of 700 ft., a surface gravity tramway is utilized to transport the ore to the yard. Two Stein trams were installed and were so arranged that they can be operated singly or both at the same time.

The surface incline has 60-lb. rails and a 36-in. gage, with an inclination varying from 30° at the top to 18° near the lower end. At the level of the motor-haulage tunnel a 300-ton ore bin gives sufficient storage to assure no delay to the motor trains. The gravity-tram skips are loaded from this bin through air-lift gates.

At the bottom of the incline a circular steel ore bin of 2,000 tons capacity gives sufficient storage to avoid delay in the operation of the tram even if railroad cars are not furnished for several hours at a time.

The skips on the gravity tram have a capacity of 12 tons, and dump direct into the steel ore bin through doors in the bottom of the skips. Railroad cars are loaded from the steel ore bin by means of air-lift gates, only 4 min. being necessary to load a 70-ton car.

One engineer at the head-house of the gravity tram can easily lower 4,000 tons of ore over this tram in a shift of 9 hr.

The brakes on the drums of these trams are tightened by dead weights, and released when the weights are raised by compressed air. When the supply of compressed air fails, the brakes are automatically applied, eliminating any danger of a runaway.

Cost of Operating Gravity Tramways

		Cost per Month
At capacity of 3,000 tons in 9 hr.:		
Labor:		
Tram engineer	$135.00	
Loaders at bins	525.00	$660.00

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Supplies:		
Compressed air	$13.50	
Miscellaneous	14.00	27.50
Repairs and Renewals:		
Labor:		
Repair man	$82.50	
Miscellaneous	12.00	94.50
Supplies		60.00
Total		$842.00
Cost per ton		$0.0094

Safety of Mining System

As many engineers have gained the false impression that a system of mining involving the use of shrinkage stopes is necessarily hazardous, a list of the fatal accidents during the 3 years under discussion may be of interest. The following is a complete list of fatalities for a 3-year period during which time more than 3,000,000 tons were mined at a cost for timbering of less than 5c. per ton:

	Number of Fatalities
In stopes:	
Falling rock	3
Other workings:	
Falling rock	1
Man fell into ore chute	1
Electrocuted	1
Asphyxiated	1
Total	7

Most of these accidents were entirely due to the carelessness of the men injured. A noteworthy feature is that not a man was fatally or seriously injured through the use or handling of explosives.

Conclusions

The system of stoping as practiced was eminently satisfactory to supplement the steam-shovel operations without injuring the ore reserves of the property through a mixing of capping with ore. It had the advantage that all ore produced was absolutely free from waste, since both stopes and development drifts were discontinued when capping was reached. The assay value of the ore produced could be regulated, and the tonnage materially increased or decreased without affecting to any extent the cost per ton.

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Interpretation of Assay Curves for Drill Holes*

BY EDWARD H. PERRY,† M. E., CAMBRIDGE, MASS., AND
AUGUSTUS LOCKE,†† M. E., S. D., CAMBRIDGE, MASS.

(New York Meeting, February, 1916)

IN the exploration of a copper deposit by drilling, obvious advantages are to be gained from a distinction between primary and secondary ore.[1] Perhaps the chief of these is the aid which such a distinction renders in determining where a given hole should stop. The copper assay, often a sufficient guide, cannot always be exclusively relied upon. Primary ores may extend downward indefinitely and may fluctuate in value independently of depth: if primary ores are consistently lean for a considerable vertical distance, there is little reason to expect that, still deeper, they will increase to the commercial grade; on the other hand, rich primary ores may persist uninterruptedly downward or may come in again below a lean interval. Secondary ores, however, have a comparatively limited vertical extent; in a large way enrichment decreases with depth and finally gives out. It follows then, that when a hole descends from profitable ore into ore of less than the commercial minimum, the practical significance of the situation depends upon whether this decrease results from a decrease in secondary enrichment or is a variation in primary content: if the copper of the ore passed through has been chiefly secondary, little is gained in the average case by extending the hole much below the point at which the commercial limit is reached; but, when copper is chiefly primary, due weight must be given to the possibility that, still deeper, the ore may again improve.

The distinction between primary and secondary ore may of course be effected through an examination of the drillings. It has been found, however, to be facilitated by curves made by plotting assays against

* This is one of a series of contributions by the Secondary Enrichment Investigation that are intended to present results of specific portions of its work in advance of the complete memoir.

† Mining Geologist.

†† Mining Geologist.

[1] In this paper, the terms *secondary* and *enriched* are used in the sense commonly ascribed to them in the expression "secondary enrichment," and refer to changes undergone by initial, *i.e.*, *primary*, constituents of an ore as the result of solutions descending from the surface. The term *ore* as here used does not necessarily signify material of commercial grade and consideration is given only to sulphide, not to oxidized ores.

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depths. In this connection, the data of several hundred drill holes in the copper districts of Ajo, Bingham, Ely, Miami, Ray, and Santa Rita, courteously furnished by officials of the local companies, have been studied in conjunction with the corresponding drill pulps and ore specimens. The accompanying curves represent holes chosen to illustrate both normal and abnormal conditions, which are respectively simple and difficult of interpretation; they are intended particularly to indicate how a presentation of the facts by their means readily permits deductions concerning secondary enrichment in copper ores.

Normal Conditions in Porphyry[2] and Schist

The ores, whether primary or secondary, that are best adapted to exploration by drilling are those in which the valuable minerals are uniformly distributed. The result of enrichment on primary material of this kind is an ore not only richer but of somewhat more variable grade, though still sufficiently uniform to be suitable for drilling. Since a condition of uniform distribution is that most commonly met in drilled deposits, it may for the purposes of this paper be regarded as normal. Though present in some other rocks, it is most likely to occur in porphyry or schist.

The significant features of curve-shape are chiefly concerned with the abruptness of the change in copper content. In particular, a jagged profile is to be contrasted with a smoother one, and, according as they possess one or the other of these characters, curves representing normal conditions fall into two principal groups:

I. That in which the tenor undergoes no abrupt change throughout the sulphide part of the hole (*e.g.*, curves 1, 2, and 3).

II. That in which the lower part of the curve has the characteristics of group I, while the upper part is richer and is marked by peaks (*e.g.*, curves 4 to 8).

Primary ore commonly yields a curve of mild profile. A curve of secondary ore is commonly peaked, and, as might be expected, the more emphatic the enrichment, the higher and rougher are the peaks, and the more characteristic is the form of the curve. A curve, then, of group I presumably means primary ore, while a curve of group II presumably means primary ore below secondary ore. The curves noted as illustrating these two groups are, in fact, exactly what their forms thus suggest.

In group I, curves 1 and 2 well exemplify, by the smoothness of their profiles, the comparatively uniform tenor of primary ores under conditions defined as normal. They also illustrate a not unusual tendency of primary ore to decrease gradually in value with depth. Curve 3 indicates a somewhat less constant grade of ore, yet sufficiently uniform to

[2] The term *porphyry* is used in the popular and not in the strictly technical sense.

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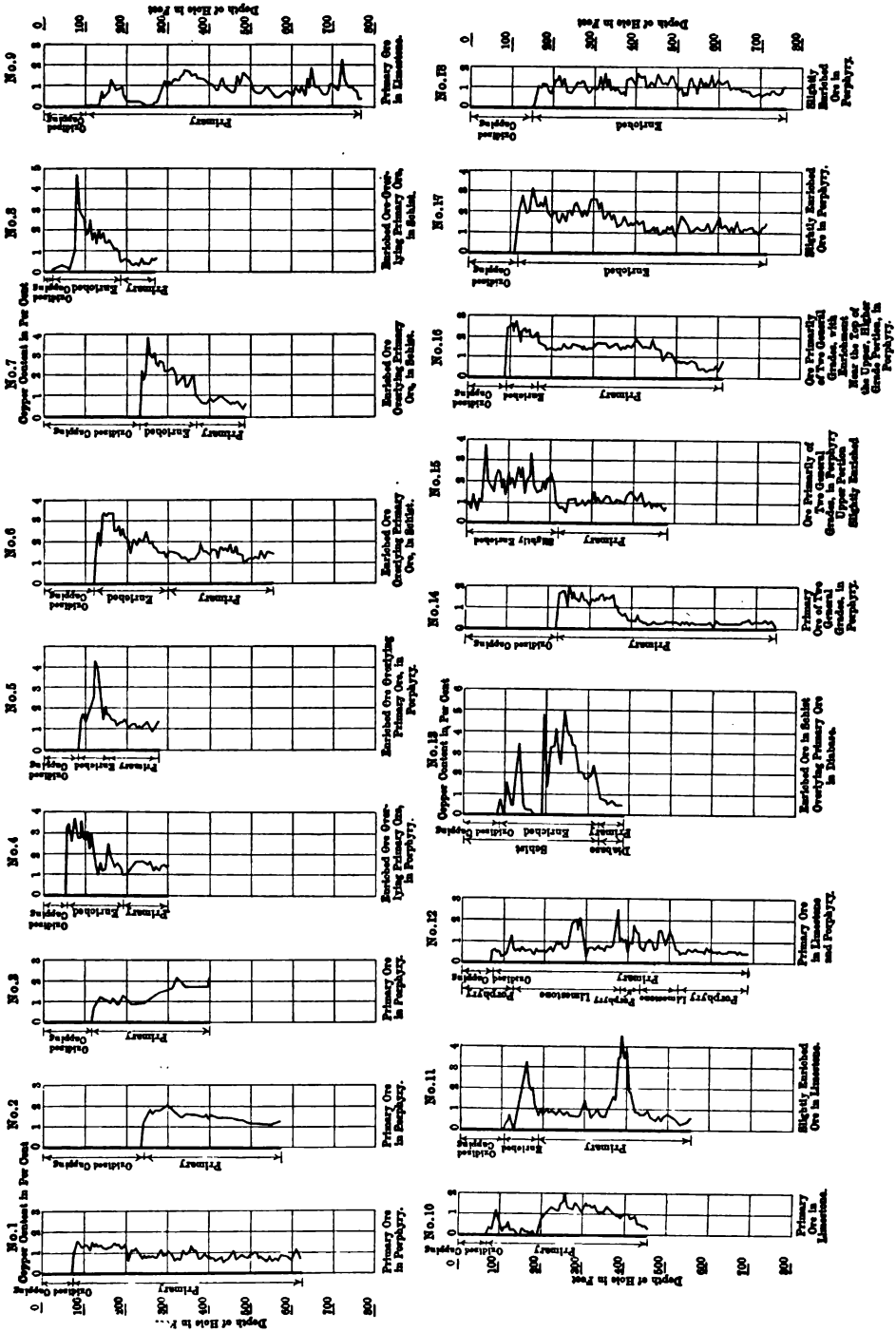

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place the curve in group I; in any case, there is nothing in its shape to suggest enrichment.

In group II, curves 7 and 8 bring out strikingly the effect of strong enrichment on low-grade primary stuff; in curves 4 and 5, though the primary ore was of better grade and the ratio of enrichment somewhat lower, the effect of enrichment is clearly shown by the distinct difference in character as well as in tenor between the upper and lower parts of the curves; in curve 6, good primary ore, not very strongly enriched, yields a profile less strikingly different in its two parts, though even here the situation is plain. In all these five curves, the conditions are to be regarded as normal.

It is fortunate that these two types of curves, which are generally easiest of interpretation, are the ones most commonly encountered.

Disturbing Influences of Other Rocks

An interpretation of ore character, however, by means of curve shapes alone is often beset with difficulties. Departures from normal conditions may be encountered in porphyry or schist but are met more frequently in other rocks. They are most likely to be found in limestone, which is fairly common in deposits explored by drilling, and which, on account of the erratic nature of its mineralization, often introduces disturbing factors. Various kinds of difficulties that are experienced find illustration in curves 9 to 13, which may be classed as still another group:

III. That in which decided irregularities occur once or more, without systematic relation either to each other or to the top of the hole.

Curves 9 and 10, representatives of a type which is common, are of primary ore in limestone. If they were in schist or porphyry their moderately peaked character extending to the bottom would suggest deep and rather mild enrichment.

Curve 9 serves to illustrate also how the copper assay alone might lead to an erroneous decision as to the depth at which the hole should be bottomed: on the basis of copper assay alone, the hole might have been stopped at a depth of about 250 ft. or at a depth of about 600 ft.; if the decline in tenor just above either of these horizons had been due to a playing out of enrichment, bottoming at one or the other of these depths would have been justified; but since the ore is in fact all primary, it was wise to sink the hole deeper and actually there is no indication that the hole is yet deep enough.

Curve 11 is of slightly enriched ore in limestone. The existence of two peaks widely separated is not typical of enrichment in schist or porphyry, and suggests primary ore of erratic or bunchy distribution and this in turn suggests limestone. The two peaks in the curve might, for example, coincide with two of the limestone beds that were

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especially favorable for primary ore deposition. As a matter of fact, were the effect of enrichment removed from this curve, its shape would be only slightly changed. This situation well illustrates the impossibility of detecting enrichment in limestone from the shape of the curve alone.

Curve 12 is of primary ore in alternating porphyry and limestone. It has the appearance of being a typical curve of primary ore in limestone, but there is no way of determining from its shape whether or not enrichment is involved in any part of it.

Curve 13 is of enriched ore in schist overlying primary ore in diabase. Altogether, the curve looks like one in porphyry or schist with irregular enrichment above primary ore; the barren stretch near the middle of the hole might be variously interpreted as the result of leaching or as due to primary poverty, and the small length of low grade at the bottom might suggest a return to primary conditions. As a matter of fact, the low grade at the bottom is due to the presence of the diabase into which the enriching solutions were unable to penetrate effectively, and the barren interval is the result of oxidizing influences, which have entered obliquely or laterally, and have thus undercut sulphides lying above.

Erratic Conditions in Porphyry and Schist

Abnormalities which are occasionally present in deposits contained in porphyry or schist may bring about conditions as difficult of interpretation as those due to the presence of limestone or of other disturbing rocks, or they may yield curves which, on the basis of shape, would fall in group II or even in group I, but which in reality owe their shape to causes other than those normally determining the profiles characteristic of groups I and II respectively. Curves 14 to 16 afford examples of such erratic or abnormal conditions.

Curve 14 is of primary ore in porphyry. Viewed broadly, it suggests enriched ore over primary ore; yet its upper and richer part lacks the strongly serrated outline usually found to accompany enrichment. Indeed, each of the two parts, taken independently, looks primary, and this is actually the case, the drop in tenor coinciding with the entrance of a porphyry of a different variety which, in the district in question, carries less copper.

Curve 15 is of ore in porphyry, only slightly enriched, and representing two primary grades, the poorer underlying the richer. The primary copper of the poorer is in chalcopyrite; that of the richer is in bornite as well as chalcopyrite. On the basis of shape, the curve is an especially good example of group II, which would ordinarily signify enriched ore overlying primary ore; yet, in this particular case, the effect of enrichment on the shape of the curve is only to accent the relief of the peaks in its upper, richer part.

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Curve 16, of ore in porphyry, might be thought to indicate deep enrichment giving way to primary ore near the bottom. As a matter of fact, enrichment is confined to the upper part, where the irregularities are most accentuated; below the point, at a depth of about 170 ft., where the tenor suddenly steps down and the curve becomes smoother, the ore is wholly primary, growing distinctly leaner near the bottom.

Curves 17 and 18 are in porphyry and are slightly enriched throughout. They are not to be regarded as abnormal; they represent a normal condition which yields a somewhat erratic curve. Nothing in their shape, however, distinguishes them from curves of primary ore in limestone or even perhaps of primary ore of somewhat variable grade in porphyry or schist.

The causes of the lack of primary uniformity encountered in some ores in porphyry and schist are various and often obscure. Low-grade pervasive primary mineralization is in certain cases localized by the contacts between intrusive rocks and the rocks which they invade; vagaries in the distribution of sulphides may therefore be concerned with a neighboring contact. The presence of a mineralized dike, or of a vein, or an unaccountable increase of disseminated primary sulphides, may, for example, produce irregularities that look like the peaks of an enrichment curve. The fact, moreover, that, in the districts involved, enrichment always takes place through the replacement of one sulphide by another, and that it acts selectively, in such a way that bornite yields to its effects more readily than chalcopyrite, and both of these more readily than pyrite (to name only the three primary sulphides most commonly involved), causes peaks due to primary richness to undergo a misleading exaggeration, as is illustrated in curve 15.

Requisites for Correct Interpretation

After all, there are a number of varieties of knowledge, not yielded by curves, which are essential to an intelligent diagnosis of conditions. The study of curve shapes must be made with an understanding of the geologic habits of the district. Not uncommonly there is a decrease in the grade of primary ore with depth which may involve an entire district or affect only a certain portion; it may be related to the position of a contact or dependent on some other influence, but whatever its cause, it must be taken into consideration if the curves are to be correctly understood. A knowledge of the kind of rock traversed is invariably required and there must always be enough study of drillings and of other specimens to yield an intimate acquaintance with the district habits.

Such studies are facilitated by the microscope. Thin sections are sometimes necessary for the identification of the rocks and determination of the alterations they have undergone. Polished surfaces of opaque

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minerals are often indispensable for the recognition of enrichment and and of its character. Powders, such as drill pulps, can be investigated by the microscope, or by the binocular, a magnetized needle aiding in the separation of magnetite from chalcocite which, in the powdered form, it closely resembles. It is often advantageous to embed powders in sealing wax, in order that polished surfaces of their constituent grains may be prepared for microscopic examinations.

Limestone—which is especially important to recognize on account of its disturbing influences—is sometimes difficult of identification in pulp form because of intense alteration. The common test of effervescence with acid may, of course, be ineffectual, because of the frequent destruction of the carbonate by the alteration. A familiarity with the conditions or character of limestone alteration in the district will usually serve in separating this rock from porphyry. The limestone, however much altered, is likely to have a characteristic aspect, due, for example, to certain habits of sulphide aggregation or of softness.

Altogether, the usefulness of a study of curve shapes will vary inversely with the amount of additional knowledge needed for their interpretation. Should the deposit or district be erratic in its habits, curves may not be very intelligible. But should the district possess a uniformity or regularity in its general geology and its mineralization, the curves are likely to be very significant and to constitute an important short-cut to an understanding of the ore distribution.

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An Electro-Hydraulic Shovel

BY FRANK H. ARMSTRONG,* B. S., VULCAN, MICH.

(New York Meeting, February, 1916)

ALL the mining machinery of the Penn Iron Mining Co. has been operated by electric power for several years and when another shovel for stockpile loading was required the advantages of an electric shovel were naturally considered. After considerable study, serious objections suggested themselves the use of a shovel operated directly by electrical apparatus by reason of the complicated control, severe service on the motors

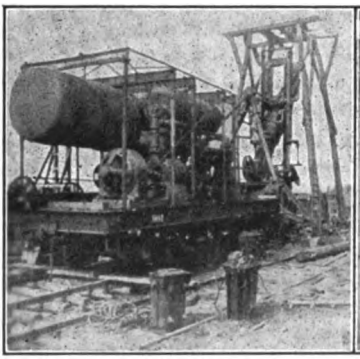

FIG. 1.—ELECTRO-HYDRAULIC SHOVEL UNDER CONSTRUCTION.

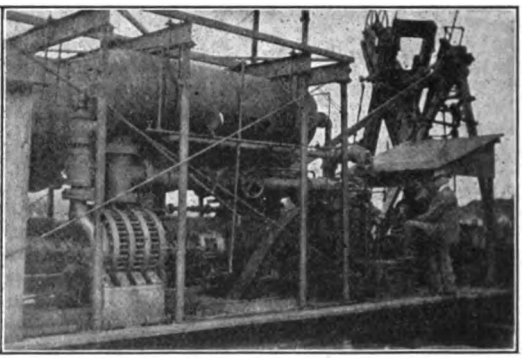

FIG. 2.—THE SHOVEL AS IT WAS OPERATED THE FIRST YEAR.

and the heavy surges of current required from the line. It was finally decided to construct an electro-hydraulic shovel using water under pressure to perform all the necessary operations except that of propelling, for which a separate motor was best suited.

A car body with boom, dipper handle and dipper of late design, but without any of the steam machinery, was purchased. A motor-driven centrifugal pump, a pressure tank, an air tank, a small air compressor and water cylinders with plungers, pistons and valves, were installed in place of the steam equipment. Fig. 1 shows this shovel under construction and Fig. 2 shows it as it was operated during the first year, the summer of 1914. The large tank at the back of the car, in Fig. 1, carried air under

* Mechanical Engineer, Penn Iron Mining Co.

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a pressure of 275 lb. The smaller tank, seen best in Fig. 2, is the water-pressure tank into which the pump discharged and from the bottom of which the water was taken by the main header to the valves. Back of the water-pressure tank is another tank, the end of which can be barely seen in Fig. 2, used as a suction tank, which received the water exhausted from all the cylinders and from which the pump drew its supply.

The shipments of ore during the summer of 1914 were so small that the new shovel was not given a thorough test, but during the summer of 1915 the shovel requirements were more favorable. It was soon found that the tanks and the air compressor could be dispensed with, since a sufficient capacity could be obtained without the use of compressed air, by reason of the fact that the capacity of a centrifugal pump increases rapidly as the head decreases. The removal of the tanks improved the

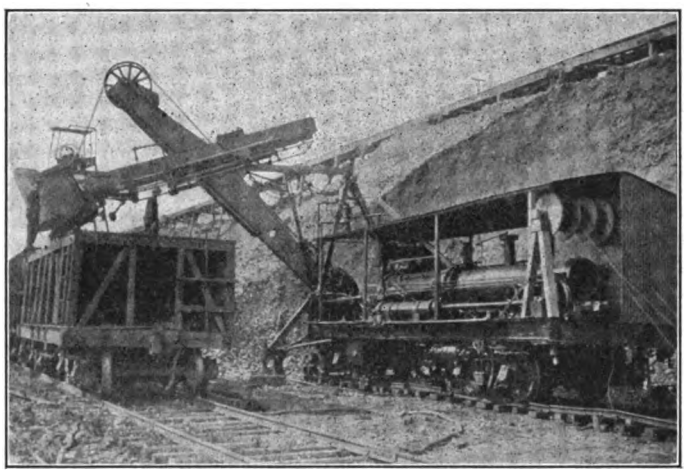

FIG. 3.—ELECTRO-HYDRAULIC SHOVEL WITH COMPRESSED-AIR TANKS REMOVED.

appearance of the shovel as well as lowered the center of gravity. Fig. 3 shows the shovel after the changes had been made.

The dipper is hoisted by means of a large cylinder and plunger. The double hoisting ropes pass around two sheaves at the outer end of the plunger. One end of these ropes is anchored to the front flange of the hoisting cylinder, while the other end is fastened to the dipper. The dipper travels 2 ft. for every foot the plunger travels. The cylinder is single acting, since the weight of the dipper pulls back the plunger when the valve is open to exhaust.

The swinging of the boom is effected by means of a double-acting cylinder with a piston and rod extending through each cylinder head, with a sheave at each end of the rod. The swing circle at the base of the

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boom is moved by a rope, the middle of which passes around the front end of the swing circle and the ends go around the sheaves on the ends of the rod and fasten to the car body.

As shown in Figs. 3 and 5, the thrusting is done by the piston rods of four thrusting cylinders directly connected to the dipper handle. The four cylinders give a perfect balance around the shipper shaft. The piping to the thrusting cylinder is made with swivels and sleeve joints, so that the dipper can be raised or lowered, or the boom swung either way or raised without putting any strain on the piping.

The shovel is operated entirely by one man as shown in Fig. 4. His right hand does all that the craneman on a steam shovel does. He pushes

FIG. 4.—ALL SHOVEL OPERATIONS CONTROLLED BY ONE MAN.

the lever from him to thrust out and pulls it toward him to bring in the dipper. His hand if dropped a trifle strikes the button shown below the hand and trips the dipper door. His left hand operates the valve for the hoist; forward to hoist, toward him to lower. His feet operate the swing valve by pushing on levers that are interconnected. By pushing on his left foot he swings the boom and dipper to the left, by pushing on his right foot he swings them to the right. When the feet are even, the valve is in the stop position to which the valve is brought automatically

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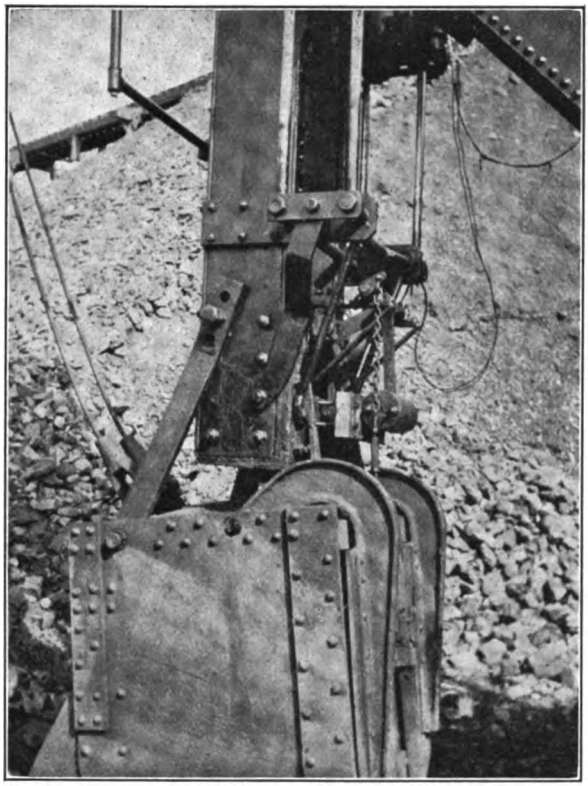

FIG. 5.—DIPPER DOWN, WEIGHT CAUGHT.

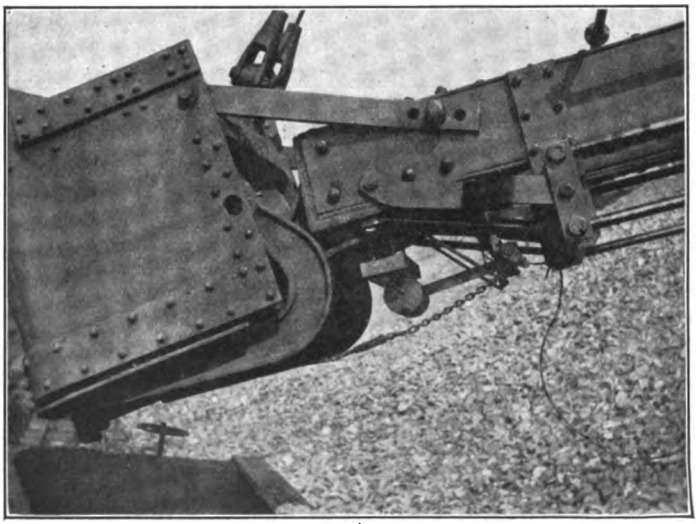

FIG. 6.—DIPPER OVER CAR.

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by a centering device. The handle of the controller for the motor which propels the shovel is seen under his right elbow.

Considerable manual effort is avoided by the solenoid tripping device for the door of the dipper. This is shown in Figs. 5, 6, 7 and 8. A weight on a long arm suspends from a horizontal shaft and tends to hang vertically, being parallel to the dipper handle when the dipper is down. A small catch holds the shaft from turning as the dipper is hoisted, and when the dipper handle is horizontal the weight is ready to fall as soon as the catch is tripped. On the same horizontal shaft is a short arm the end of which is connected by a chain to the latch of the dipper door. When the weight falls, the jerk with the leverage which the weight exerts upon the short arm pulls the chain, and trips the dipper door. The

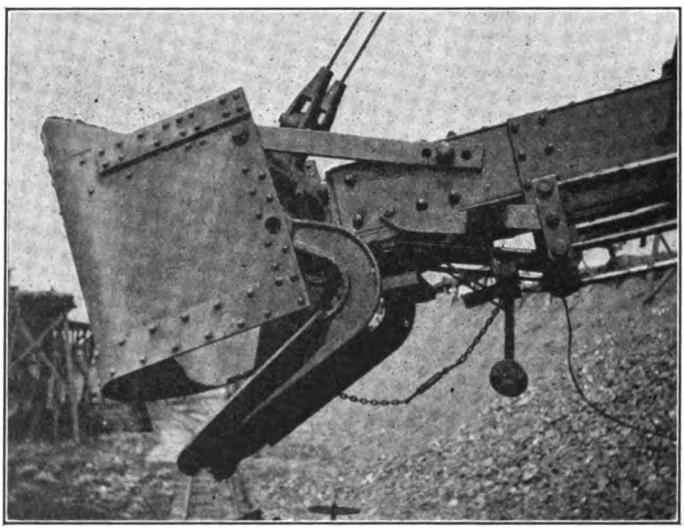

FIG. 7.—WEIGHT RELEASED, DIPPER DOOR OPEN.

small catch is released by a solenoid operated by the button under the operator's right hand. Fig. 5 shows the dipper down with the weight caught; Fig. 6 shows the dipper over the car with the weight ready to fall; Fig. 7 shows the weight tripped and the door open.

The average speed of the machine is between three and four dippers a minute. Repeatedly, 3,000 tons have been loaded in a 10-hour day, although the railroad service required shifting every two or three cars. Much more could have been loaded if good car service could have been obtained. With some slight changes it will be possible to increase even this speed considerably.

The power required and the speed of the shovel are shown in the chart, Fig. 9, from a recording wattmeter.

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This shovel has few moving parts as compared to either a steam shovel or a straight electric shovel. There are no gears, clutches, brakes or

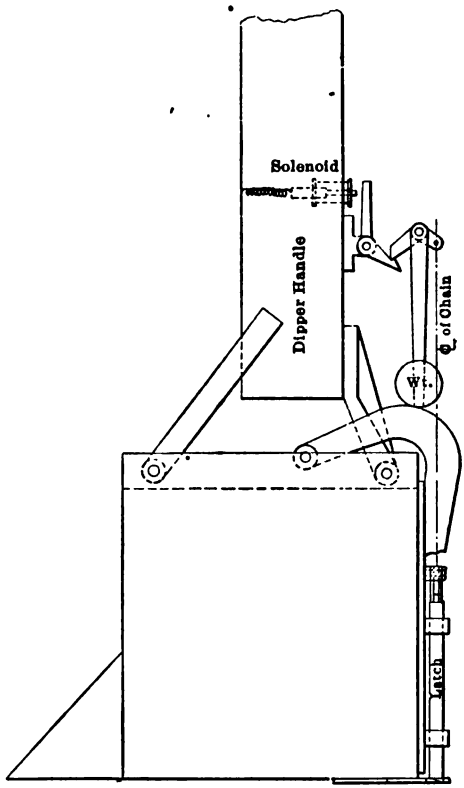

FIG. 8.—DIPPER-DOOR TRIP.

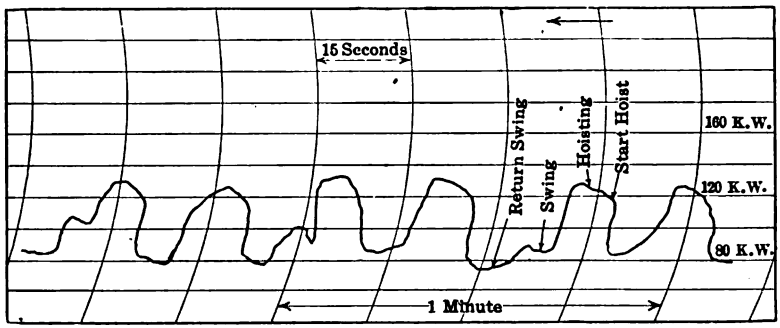

FIG. 9.—WATTMETER RECORD SHOWING POWER CONSUMPTION AND SPEED OF SHOVEL OPERATIONS.

drums. The few moving parts, except the motor and pump, move very slowly. The pump gives no trouble since the water is clean and mixed with a lubricant. The leakage is small so that an hydraulic oil that will

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not freeze can be used in winter. The only noise is the hum of the motor and a slight singing of the water as it rushes through the pipes and valves.

The advantages are that it is a one-man shovel not requiring a craneman or a fireman. The current pull from the line has no large peaks and when the shovel is not in operation the current can be shut off entirely and the pump stopped. There is very little to wear out except the dipper lip and the packing. The control of the movements of the shovel is smooth and accurate. If it is desired to raise or lower the boom for rail transportation it can be done in a few minutes by resting the dipper on the ground and either thrusting or the reverse. It is practically noiseless and clean.

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Tests on Various Electric Motor-Driven Equipment Used in the Preparation of Anthracite Coal

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(New York Meeting, February, 1916)

IN the past, steam engines were used in practically all cases for driving the machinery in and about an anthracite breaker, and hence few or no accurate data were available as to the power requirements for starting and operating the individual machines, which make up the complete equipment. It was impossible to segregate the power necessary to drive each individual machine, first, because the equipment was driven in groups, and second, because it was impracticable to obtain graphic, integrated, or instantaneous records of the horsepower used.

As a result of these conditions, it was found, when individual electric motors were applied to the various machines, that more accurate and complete data were needed in order to provide motors of the proper power and design for the particular services to be rendered.

In order to obtain the desired information, the writers made and collected a number of tests, from time to time, on the individual motor drives in electrically operated breakers, and as these tests have been of great value, it occurred to them that a paper giving the results might be of service to other engineers interested in similar installations. Table I shows the results of these tests.

All the motors tested were of the three-phase, 60-cycle, 440-volt, induction, squirrel-cage type, except a few, which, on account of size and starting conditions, were of the wound-rotor type.

The instruments used in making these tests were: An integrating watt-hour meter, a graphic ammeter, and a graphic voltmeter. These instruments were connected in each motor circuit, and a record of a day's run of 9 hours was obtained from each machine.

From these tests, the running light load, the average all-day load, and the instantaneous peak readings in amperes, volts, and kilowatt input, were obtained. By means of the characteristic curves obtained from these motors at the factory, the power factor, the efficiency, and the horsepower output were calculated for each particular test. Starting tests were made by the brake-arm method, in which the torque in pounds was measured directly on a spring balance.

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TABLE I.—*Tests of Electric Motors, Etc., Employed in Anthracite Breaker*

Item No.	Apparatus	Motor H.P.	Motor R.P.M.	Starting Torque Ft. Lbs.	Starting Torque % F.L.T.	Average All Day Test Load P.F.	Average All Day Test Load Eff.	Average All Day Test Load H.P. Output	One Min. Peak % F. Load	No. Load Test H.P. Output	Mechanical Data. Decks No.	Decks Size	Ecc. Displace	Ecc. R.P.M.	Eccentric Ecc. Shaft	Eccentric Stroke	Remarks
Shakers																	
1	Lump Coal Shakers	20	900	100	85	83	86	14.5	115	7.1	3	1-5"x12" 2-4"x12"	120°	178		6"	
2	Counter Mud " #1	20	"	85	73	80	85	12.4	76	8.2	3	5"x18"	120°	156		"	
3	" " " #2	20	"	72	61	79	84	11.5	94	8.0	3	1-5"x18" 2-4"x18"	120°	163		"	
4	Main "	20	"	135	115	80	85	12.4	82	10.0	2	5"x16"	180°	163		"	
5	Counter Main " #1	20	"	68	58	74	82	9.5	100	5.5	3	6"x18"	120°	163		"	
6	" " " #2	20	"	80	68	74	82	9.3	70	4.3	3	6"x18"	120°	163		"	
7	Penay " #1	30	"	90	51	70	80	10.8	44	7.2	4	5"x24"	90°	163		"	
8	" " #2	30	"	135	77	78	84	15.2	58	10.8	4	5"x24"	90°	163		"	
9	Washery " #1	20	"	53	45	64	77	6.4	28	4.3	2	5"x18"	180°	143		"	
10	" " #2	30	"	143	82	87	86	27.4	115	22.0	6	5"x18"	120°	163		"	Double Set
11	" " #3	30	"	81	46	81	85	17.3	75	8.2	4	6"x18"	90°	163		"	
12	" " #4	30	"	144	82	81	85	17.3	88	10.8	4	6"x16"	90°	163		"	
13	Tailings "	20	"	85	73	74	82	9.3	50	8.0	3	1-5"x18" 2-4"x18"	120°	163		"	
Rolls											R.P.M.	SIZE	Sq. Ft. Roll Surface	R.P.M. Rolls			
14	Main Rolls	35	900	226	110	85	86	25.0	120	17.5	763	40"x36"	40	73			
15	#2 "	20	"	171	146	87	75	5.3	65	5.0	1070	21"x36"	21.2	152			
16	#3 "	20	"	225	192	57	75	5.5	87	3.2	670	21"x36"	21.2	95			
17	#4 "	20	"	126	108	35	57	2.2	11	2.2	1100	21"x36"	21.2	156			
18	#5 "	20	"	108	93						1060	24"x18"	10.2	156			
19	#6 "	20	"	135	115	63	78	6.1	45	4.3	980	24"x24"	12.5	156			
Conveyors											SCRAPERS			LINE			
20	Main Conveyor	150	514			72	82	87	75	18	No	Area	Dis. Bet.	Length	Weight	SPEED F.P.M.	
21	Surface "	50	900			62	70	11.5	80	8	134	33 Sq. Ft.	48"	232	97	81	
22	Rock "	(20)	"	30	25.5	87	75	5.3	27	5.3	72	3.3 " "	45	118.5	42	58	
23	Tailings "	(10)	"	27	47	63	80	3.9	39	2.1	114	1.2 " "	27	119	0	80	
24	Cross "	(10)	"	23	40	63	80	3.9	39	1.8	42	1.2 " "	27	39.5	13'	120	
25	Refuse "	(20)	"	13	11.1	45	68	3.2	16	3.2	64	1.2 " "	27	37.5	0	114	
Elevators											65	1.2 " "	27	73	0	114	
											BUCKETS			LINE			
26	Bottom Lift Main Elev.	(30)	900	22	12.5	40	58	3.4	12	3.4	No	Cu. ft.	Dis. Bet.	Length	Weigth	F.P.M.	
27	Top " " "	(30)	"	54	31	74	82	12.8	96	11.0	66	2.6	27"	68		162	
28	Bott. Lift Lip Screen "	(20)	"	19.6	16.7	53	74	4.5	23	3.2	59	2.6	27"	58.5		120	
29	Top " " "	(20)	"	20	17.0	45	68	3.2	70	3.2	92	1.6	27"	65		110	
30	Tailings "	(20)	"	67	57	53	75	4.8	24	4.8	76	1.6	18"	51		108	
31	Buckwheat "	(20)	"	135	115	Not in use					115	1.6	27"	78		160	
Miscellaneous														43		160	
32	Upper Jigs	(50)	900	90	31	73	82	21	42	3.6	4 Jigs running						
33	Lower "	(50)	"	144	49	77	85	29	59	4.2	" "						
34	Dust Fan	75	"			82	85	41	55		220 r.p.m. 6'-9½" disk x 4'-8" face Fan						
35	Empty Car Haul	10	"	42	74*	83	85	8.5	85	6.8	4 cars 2400# each: {180' ¾ pulleys .. 1.33' rise: {80 F.P.M.						
36	Loaded " "	20	"	58	80*	90	78	15	150	7.7	4 " 8058# " {2.12" " " 7.44 " : {chain Speed						
37	Washery Pump	75	"			90	90	61	81		120' head 1800 R.P.M.						
38	Rock Pulverizer	75	"			71	84	52	130	14	Capacity 40 Tons per hr.						
39	Pickers	(5)	"	10	34	76	78	2.9	58	2.9							

* These figures apply only when haulage chain is free from ice. The results are based on trip tests, not all-day tests. NOTE.—Revolutions per minute of driven units figured at synchronous speed of motor. Voltage at motors, 415. Horsepower ratings inclosed with O are for back-geared motors. Key: F.L.T. = Full-load torque; P.F. = power factor; Eff. = efficiency; F. load = full load.

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Unusual care was taken in making these tests and in calculating the results; therefore, the writers feel that they are reliable for all practical purposes.

As the characteristic curves on the various machines differ, graphic charts taken under actual operating conditions, the results of the tests, and the mechanical data pertaining to the machine driven, follow in proper order.

Shakers

From Table I, it will be noted that tests were made on shakers of various sizes of the two-, three-, and four-deck types. The eccentric

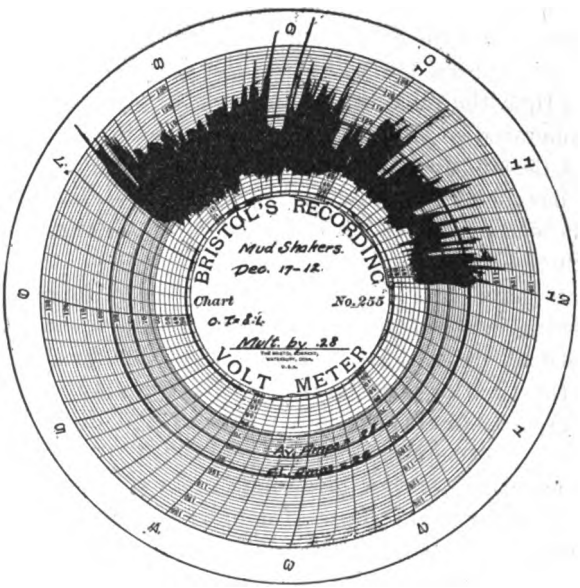

FIG. 1.—GRAPHIC AMMETER CHART SHOWING LOAD ON SHAKER SCREENS.

displacement is divided evenly in order to balance up the load, and the shakers have a 6-in. stroke. There is a slight difference in the speeds at which these various shakers are operated, the speed of the eccentric shafts ranging from 134 to 163 r.p.m.

From the graphic ammeter chart, Fig. 1, it will be noted that the characteristic load on the average shaker screen is a fluctuating one, the fluctuation varying in direct proportion to the degree of unbalancing in the various decks. This unbalancing may be caused by several factors, which are practically impossible to control, viz., unequal quantities of coal on the various decks, unequal weights of the decks, unequal friction in bearings of the suspension rods which support the decks, etc.; also,

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a fluctuation in the load may be caused by improper spacing of the eccentrics.

Theoretically, the load on each deck during a cycle of operation conforms very closely to a crank-effort curve as obtained on a steam engine; so, when the load curves of the cycle of operation of a complete shaker are superimposed, the resultant load curve is undulating. This variation in load would tend to show a considerable fluctuation on a graphic ammeter chart where the eccentric spacing is from 180 to 120°, decreasing as the spacing is decreased and the number of decks increased.

The condition which arises in connection with the load obtained on a two-deck shaker when the eccentrics are spaced 180°, deserves special mention. The cycle of operation of the decks when they are driven from the eccentrics spaced 180°, gives the best condition for minimum vibration of the shakers and the maximum variation of load on the motor. If the eccentrics are spaced 90° to obtain the minimum variation of load on the motor, then the maximum vibration is obtained on the shakers. If they were operated in this manner for any length of time, they would set up a very destructive vibration in the surrounding timbers of the breaker. It has been suggested to operate these shakers with 180° spacing on the eccentrics, and install a flywheel on the motor shaft to compensate the load fluctuations.

The writers have had occasion from time to time to investigate troubles occurring in connection with the operation of shakers, and have found in almost all cases that they were due to unequal spacing of the eccentrics at the time of installation; to the attendant's tightening the bearings on the decks or the eccentric straps; to the improper lubrication of the bearings and eccentrics, etc.

Tests and experiments were also made in an endeavor to reduce this fluctuation in load by making use of a flywheel, and also by attaching springs to the various decks; but the results obtained showed that, except possibly in the case of flywheels on two-deck shakers, conditions were not bettered sufficiently to warrant the expense of installing the additional apparatus.

Rolls

The rolls used in the tests and described in Table I were of the manganese-segment type with cast-iron spiders, excepting No. 4 which was of the chilled-tooth type.

The following are the sizes of coal treated by the several pairs of rolls:

Main rolls, Lump to grate coal;
No. 2 rolls, Grate to egg coal;
No. 3 rolls, Grate to egg coal;
No. 4 rolls, In the washery—breaking coal from refuse dump down to egg and below;

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No. 5 rolls, In the washery—breaking coal from refuse dump down to chestnut;
No. 6 rolls, In the washery—breaking coal from the refuse dump from chestnut to buckwheat.

The sizes and speeds of these various rolls are found in Table I under "mechanical data."

Since much more trouble has been experienced with the individual motor drive used in connection with main rolls than with any of the other individual drives, the writers have made exhaustive tests and investigations on this particular equipment. Trouble developed with these machines soon after the motors were installed, on account of enormous peaks suddenly thrown on the motor through the crowding of the rolls due to irregularity of feed. This developed into a serious matter, as this

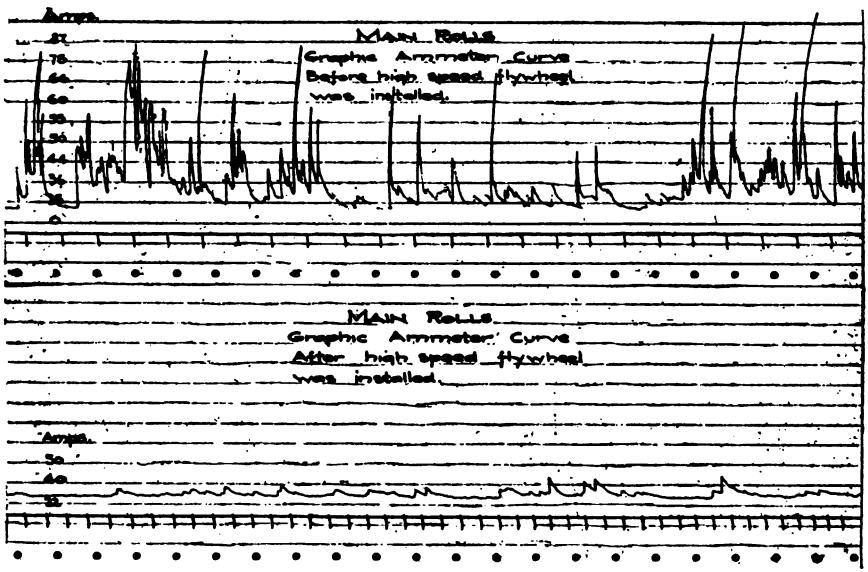

FIG. 2.—GRAPHIC AMMETER CHART SHOWING LOAD ON MAIN ROLLS.

feed reached sufficient proportions to stop the motor, and in time roasted the insulation so badly as to necessitate rewinding. Yet this, in itself, was really a minor matter when compared to the annoyance and the number of delays in the operation of the breaker. In order to locate this trouble, graphic ammeter charts were taken with a special fast-reading instrument, samples of which are shown in Fig. 2.

At this time a 2000-lb. solid cast-iron flywheel was mounted on the roll shaft, and run at about 96 r.p.m. Upon checking up the inertia of this wheel, it was found to be practically valueless at this speed.

From Fig. 2 it will be noted that the load on these rolls varied greatly so that the peaks at times reached the breakdown point on the motor.

In order to rectify this trouble, it was found advisable in some cases

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to design a special feeder to deliver the coal to the rolls more uniformly. A type that was found to work well was shaped like a paddle-wheel, and was installed in the chute leading to the rolls. The feeder was revolved by means of a belt driven from a pulley on the motor shaft or from one of the countershafts. Where picking tables or conveyors are used, the feed is sufficiently regular for good operation without the paddle-wheel feeder. After the coal passes through the main rolls, the flow is sufficiently regular on the other rolls for all practical purposes.

In designing special flywheels to take care of the large peak loads thrown on the main rolls, a careful study was made of the graphic ammeter charts obtained from the tests on this equipment. From these peaks it was found that certain conditions had to be taken care of in order to smooth out the curve, and with this information at hand, a special high-speed flywheel was designed and built up from boiler-plate iron.

After this new wheel was installed, other charts were taken with the same high-speed graphic ammeter. One of them is shown in the lower half of Fig. 2. The comparison of the charts shows conclusively that the peak conditions were nicely taken care of by the flywheel. These rolls have not been blocked since it was installed.

The size and capacity of this special wheel were calculated as follows:

By measuring and calculating several peak loads, it was found that the horsepower-seconds of these peaks were about 100. Since the total horsepower-seconds in the old flywheel at 160 r.p.m. was only 95, and for a 10 per cent. slip in the motor there were only 18 hp.-sec. available, and since a watt-hour meter showed that the average load was less than half load on the motor, the inadequacy of the flywheel was evident. It was therefore decided that it would be necessary to replace the old flywheel with a new one so designed that it would carry these peaks without slowing down to such an extent that the motor would be seriously overloaded.

The new flywheel was calculated from the following formula:

$$\text{hp.-sec.} = \frac{WV^2}{2g \times 550}$$

Where W = the weight of the wheel in pounds.

V = the velocity at the radius of gyration in feet per second.

By substituting $\frac{2\pi RS}{60}$ for V we have

$$\frac{WV^2}{2g \times 550} = \frac{W\left(\frac{2\pi RS}{60}\right)^2}{2g \times 550} = \frac{WR^2S^2}{3{,}200{,}000}$$

Where R = radius of gyration in feet.

S = speed in revolutions per minute.

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In the first place it was decided to design the wheel so that it would give out approximately 160 hp.-sec. for a 10 per cent. slip or reduction in the speed of the motor.

As the horsepower-seconds in a wheel are proportional to the square of the speed, the total horsepower-seconds necessary in order to give out 160 hp.-sec. in slowing down 10 per cent. (or to 90 per cent.) speed would be

$$\frac{160}{1.00 - (0.90)^2} = \frac{160}{0.19} = 840 \text{ hp.-sec.}$$

Since it was desirable to keep the weight of the wheel as small as possible, it was found that the wheel would have to run at the speed of the motor or 900 instead of 160 r.p.m., the speed of the rolls. It was also thought that in order to make the wheel perfectly safe, it would be better to build it up from boiler iron and limit the rim speed to 10,000 ft. per minute. Substituting these values in the above formula, we have

$$840 \text{ hp.-sec.} = \frac{WR^2(900)^2}{3{,}200{,}000}$$

As the wheel was to be built up, the radius of gyration would be 0.7 of the total radius. For 900 r.p.m. and 10,000 ft. per minute rim speed, the radius of the wheel is 1.88 ft. or a radius of gyration of 1.31 ft. All of the dimensions of the wheel were thus determined with the exception of the face or thickness. Since a cubic foot of iron weighs about 480 lb., it was found that the thickness of the wheel would have to be 5 in. It is interesting to note that the new wheel weighs only 2,190 lb. or only 190 lb. more than the old wheel, but on account of its speed it delivers nine times the horsepower-seconds in dropping 10 per cent. of its speed.

Another important item for attention is the starting torques of these rolls. Table I shows under the heading of percentage starting torque, that in almost every case over full-load torque is required. The writers suggest for consideration the possibility of using roller or ball bearings in order to decrease the high starting torque.

Conveyors

The conveyor lines are of the ordinary single, chain-flight type, with the exception of the main and the refuse conveyors, which are of the mono-bar and double-chain types, respectively. The length, height, and speed of the line, also the number, area, and the distance between scrapers are shown under "mechanical data" in Table I.

Tests indicate the load to be fairly constant and the friction load very high. A graphic ammeter chart showing the characteristic load on one of these lines is shown in Fig. 3. This chart was taken on the main conveyor line which carries all the coal from the hoisting shafts to the breaker.

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There has been no trouble of any importance in connection with the motor driving this line. The size of the motor to be used depends on the elevation and length of the line, the amount of coal to be moved, and possibly on the type of conveyor.

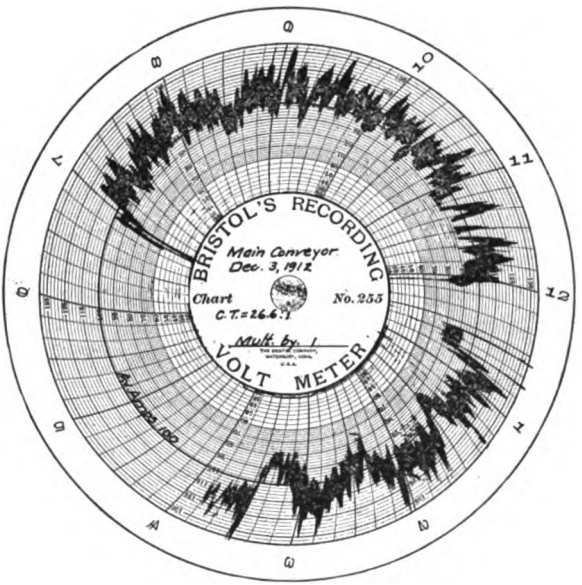

FIG. 3.—GRAPHIC AMMETER CHART SHOWING LOAD ON CONVEYOR LINE.

Elevators

The elevator lines are of the "perfect" and gravity discharge types, equipped with double chains, with the exception of the buckwheat elevator, which has a single chain. The length and speed of the lines, also the number, cubic contents, and spacing of buckets, are given in Table I under "mechanical data."

The load developed on these lines, like that obtained on a conveyor line, is practically constant. A characteristic chart of the load on one of these lines is shown in Fig. 4. This chart was taken on the top elevator which handles the coal from the refuse conveyor and carries it to the top of the washery.

No serious trouble has been experienced with the motors installed on these lines. The determination of the proper size of motor is dependent practically on the amount of coal and the height to which it is lifted.

Jigs

Jigs of the Hazleton standard, single type are operated in groups from a line shaft. Each jig is operated from this line shaft through a counter-

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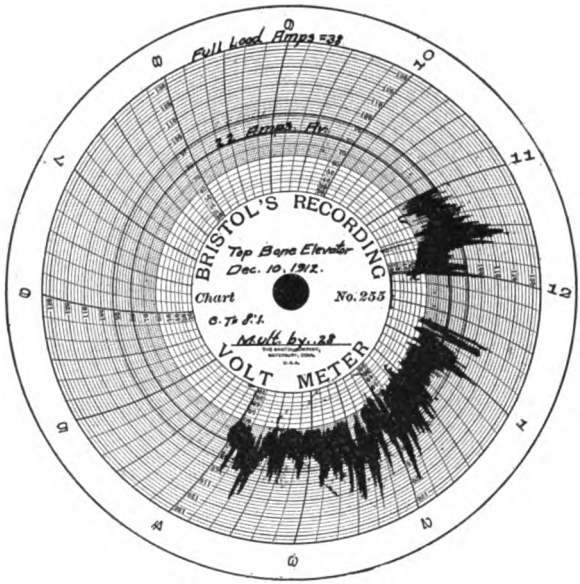

FIG. 4.—GRAPHIC AMMETER CHART SHOWING LOAD ON ELEVATOR LINE.

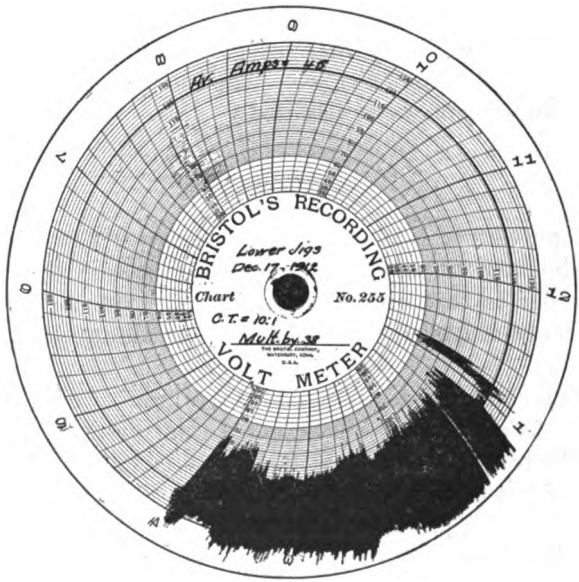

FIG. 5.—GRAPHIC AMMETER CHART SHOWING LOAD ON JIGS.

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shaft, on which are mounted the eccentrics necessary for the reciprocating motion required.

From the graphic chart shown in Fig. 5, it will be noted that the characteristic load on these jigs resembles that of the shakers in fluctuating greatly.

No serious trouble has developed with the operation of these jigs.

Dust Fan

The dust fan is of the multivane type and its size, speed, etc., will be found in Table I.

With this type of fan the load is practically constant, depending on the amount of air passing and the speed; it was thought therefore that a graphic record of its performance would be uninteresting.

Car Hauls

Car hauls consist of a single, endless, reinforced chain with 9-in. pitch equipped with knockovers to catch the car. The load is dependent on the running-light friction of the chain, the friction of the cars, the grade of the haul, the weight of the loaded car, and the speed.

For loaded cars or trains there is practically no variation in the power required.

Washery Pump

This pump is of the centrifugal type, and the capacity and head under which it operates are found in Table I. The load is constant, depending on the head and amount of water handled.

Rock Pulverizer

The rock pulverizer is of the bar-and-hammer type, and its capacity is approximately 40 tons per hour. The load on this apparatus is somewhat similar to that obtained on the rolls, viz., being made up of sharp instantaneous peaks, while the average all-day load is comparatively low.

Table II is a collection of tests made on a number of rock pulverizers now in service, from which it will be noted that under average operating conditions 17.8 tons of refuse per hour was handled, with an average consumption of power of 1.488 kw.-hr. per ton, also that the ratio of the average instantaneous peaks to the average load on the motor was 1.92. Fig. 6, a graphic chart taken on one of these rock pulverizers, probably shows the characteristic load to better advantage.

These peaks are due to irregularity in the flow of rock to the pulveri-

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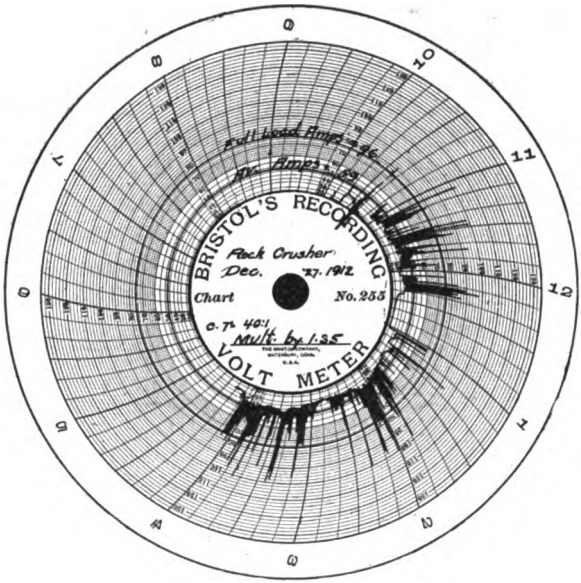

FIG. 6.—GRAPHIC AMMETER CHART SHOWING LOAD ON ROCK CRUSHER.

TABLE II.—*Tests on Rock Pulverizers*

Colliery	1	2	3	4	5	6	7	Aver.
Motor horsepower.....	75.0	75.0	40.0	75.0	75.0	40.0	75.0	.....
Motor speed, r.p.m.....	855.0	1110.0	855.0	1110.0	1110.0	855.0	1110.0	
Crusher speed, r.p.m.....	780.0	.....	.....	800.0	850.0	880.0	795.0	.....
Number of hammers.....	104.0	44.0	44.0	44.0	44.0	44.0	44.0	.....
Spacing of grates.....	2-in. rd. mesh	0.875	0.875	0.875	0.875	0.875	0.875	.....
Clearance between grates and hammers, inches.....	⅛ to ⅛	½ to ¾	½ to ¾	½ to ¾	½ to ¾	½ to ¾	½ to ¾	.....
Tests								
Rock in tons per hour.....	14.0	17.0	10.0	16.0	11.0	20.0	28.0	17.8
Input, average kilowatt.....	31.0	31.2	18.0	25.0	19.7	19.4	28.3	24.7
Output, average horsepower.	34.0	35.0	19.8	26.0	19.7	19.0	30.0	25.7
Input, kilowatt-hour per ton.....	2.22	1.84	1.8	1.55	1.79	0.95	0.93	1.488
Ratio instantaneous peaks to average load.....	1.6	3.0	4.0	1.55	1.55	2.5	2.4	1.92
Ratio instantaneous peaks to rate load.....	0.84	1.5	3.0	0.75	0.70	2.0	1.25	1.108
Average power factor.....	0.73	.....	.....	0.64	0.55	0.74	0.69	0.648
Load factor.....	0.45	0.46	0.50	0.35	0.26	0.48	0.40	0.388
Fly Wheels								
Diameter in inches.....	None	32.0	32.0	32.0	32.0	32.0	32.0	24.0
Face in inches.....	.....	4.0	4.0	4.0	4.0	4.0	4.0	.....
Rim depth, inches.....	.....	3.62	3.62	3.62	3.62	3.62	3.62	.....
Total horsepower seconds...	.....	.....	213.0	176.0	200.0	213.0	176.0	.....

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zer. The original pulverizers were driven by 50-hp. motors, which were very often loaded to their breakdown point when crowding occurred. In an endeavor to eliminate this difficulty gates were installed in the chutes leading to the pulverizer, so that the attendant could regulate the feed. The writers would not care to recommend this scheme, unless it is in charge of an experienced man, because disastrous results have followed when this machine was operated by a green man. It has been proposed to use a paddle-wheel feeder similar to that mentioned under rolls.

Conveyor lines are sometimes used to carry the rock to these pulveri-

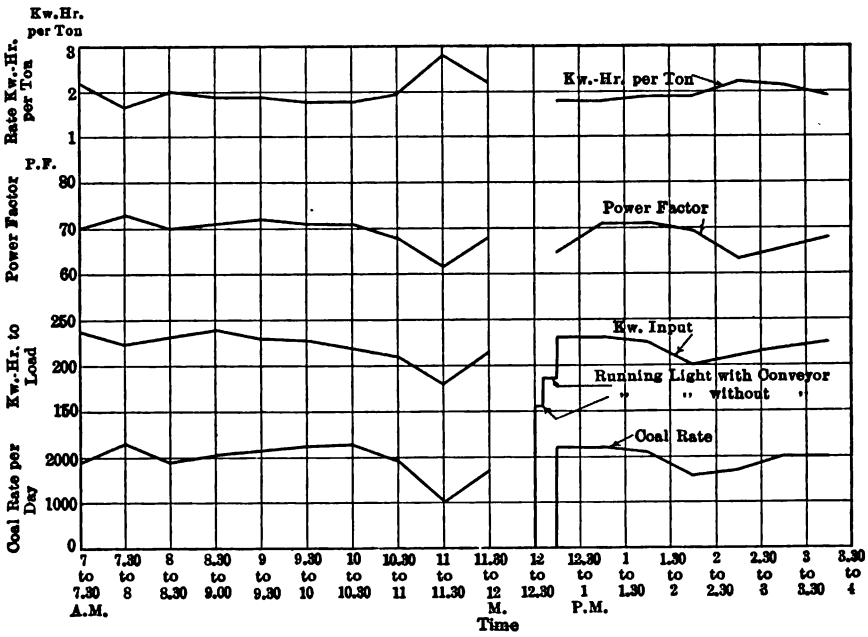

FIG. 7.—CHART OF TOTAL LOAD ON BREAKER, INCLUDING MAIN CONVEYOR, ROCK PULVERIZER, WASHERY PUMP, AND CAR HAUL.

zers, and they act as good feeders. No trouble has been experienced where they were used.

Pickers

The slate pickers are of the standard Emery type, and as the load is comparatively constant and light, and also does not have any inherent characteristic of any interest, no chart is shown of them.

Summary

Curves giving the coal rate, load on the breaker, with and without the main conveyor running, the total kilowatt input, the power factor, and the rate in kilowatt-hours per ton, are shown in Fig. 7.

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These curves, as will be noted, are all plotted over time on the base line, and the readings were taken every half hour. It is therefore easy to see how these factors vary with the rate of coal flowing through the breaker.

In Fig. 8, curves are given showing the variations in the kilowatt input, the load and the power factors in the breaker, also the efficiency and the horsepower output of the main conveyor line, with different rates of flow of the coal through the breaker.

Much could be said as to the proper type and design of motors for this service, but as it was thought to be outside the scope of this paper, it was deemed advisable not to discuss them at this time, but simply to say that they should be of heavy and rugged design.

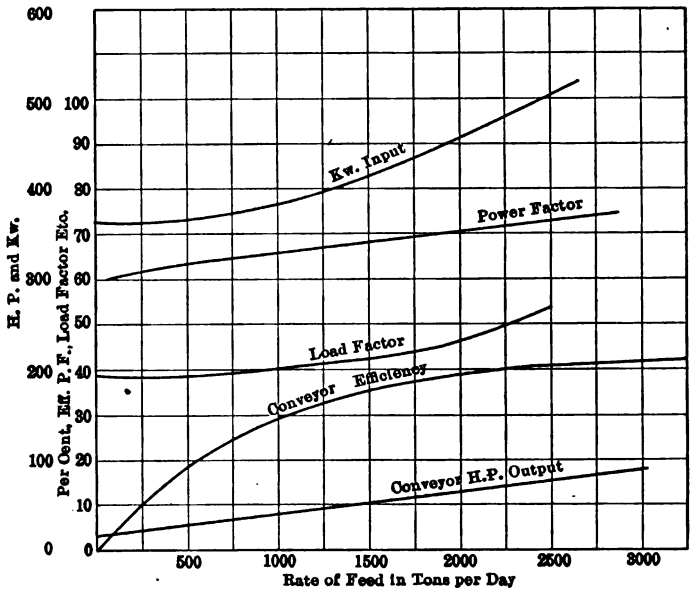

FIG. 8.—VARIATION IN THE KILOWATT-INPUT, POWER AND LOAD FACTORS OF THE BREAKER, ALSO THE EFFICIENCY AND HORSEPOWER OUTPUT ON THE MAIN CONVEYOR WITH DIFFERENT RATES OF FEED.

In conclusion, the writers wish to emphasize the importance of making a careful study of the characteristics of the machine to which the application of individual motors is to be made, since, for constant loads, the determining point in selecting the size of the machines is dependent on the heating of the motor. Then, again, machines with either high starting torque or large instantaneous peaks, or possibly both, require a motor to meet these conditions rather than for continuous capacity.

To a careful student, some of the motor ratings in the tables here given may seem a little large when compared with the tests; but this is explained by the fact that the breaker on which these tests were made was handling at the time only about 60 per cent. of its rated capacity.

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Discussion

R. V. Norris, Wilkes-Barre, Pa.—For many years the power requirements of breakers designed in the Anthracite Region have been based on calculations made from indicator diagrams taken during operation, with various sections of machinery idle; these, while giving reasonably accurate figures, gave no intimation of the peak loads which have been shown so clearly in the paper. The high-speed flywheel for breaker rolls is new and shows a remarkable saving in peak power required, besides eliminating the stalling of the rolls from overload. The figures given are so far in advance of anything previously published that they will undoubtedly be used as a basis for future breaker design. It is to be hoped that the authors will continue this investigation and extend their paper by a description of the motors found best suited for this work.

William Kent, Montclair, N. J.—I notice on p. 113 reference is made to one of the difficulties being the starting torque, and it is proposed to use roller or ball bearings in order to decrease the high starting torque.

I suggest that before the roller or ball bearings are put in, an investigation be made as to the amount of the starting torque, as shown by wattmeter measurements and as to what the starting torque ought to be, theoretically, according to the energy formula; that is, how much starting torque is required to accelerate the rotating mass in a given number of seconds up to full speed. Having secured this information it may be found the amount to be saved by roller bearings is not very great.

If it appears that roller bearings will not greatly decrease the starting torque, the next thing to be done is to have a special design of motor in which there is a field that can be made very strong at the start, by the use of extra poles or other means, and the next thing that may be done is to increase the speed from 10,000 r.p.m. to 25,000, which can be done with wire-wound flywheels, especially if the wires are made of vanadium steel.

H. M. Warren, Scranton, Pa. (communication to the Secretary*).—The discussion by Mr. Kent and the line of procedure he proposed is apparently based on the assumption that the high starting torque referred to is due, principally, to the torque required for acceleration of the flywheel; whereas, by referring to Table I, it will be noted that the starting torque of the main rolls is 110 per cent. of the full-load torque of the motor. This starting torque test was made by the brake-arm method, as stated on p. 107, and, therefore, does not include any accelerating torque, simply the starting friction of the rolls.

* Received Mar. 8, 1916.

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It is not possible to provide special field windings to increase the starting capacity of the motor, as these motors are of the three-phase A. C. squirrel-cage type.

The authors see no object in considering the use of wire-wound wheel in order to increase the speed to 25,000 ft. per minute: first, because this speed is not necessary; and second, if a higher speed were necessary, a wheel of this same type could be operated up to 18,000 ft. per minute; but because of the difficulty in maintaining proper shaft alignment in the ordinary breaker, the speed of 10,000 ft. per minute is considered more practicable than a higher speed.

K. A. PAULY, Schenectady, N. Y. (communication to the Secretary*). —One of the most aggravating problems which the engineer meets in the equipment of a new plant is that of the power required to drive the various elements. Especially is this the case if a radical change has been made in the grouping of machines requiring an accurate knowledge of the power requirements of each machine to determine the capacity of the motor driving the group. The paper by Warren, Biesecker, and Powell supplies some very much needed information regarding the power required to drive the various portions of a modern anthracite coal breaker. I am certain that the information will be welcomed by all who are interested in equipping similar breakers.

One important point which I wish to emphasize is brought out indirectly in this paper and that is the value of making tests of motor drives and studying the load diagrams with a view to making improvements where possible. The ease with which motor drives can be studied and improvements introduced is a very strong argument in favor of electrification, which is often lost sight of when considering several competitive drives.

* Received Feb. 11, 1916.

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The New Electric Hoist of the North Butte Mining Co.

BY FRANKLIN MOELLER,* CLEVELAND, OHIO

TOGETHER WITH A NOTE ON PRELIMINARY CALCULATION OF FLYWHEEL MOTOR-GENERATOR SET

BY C. D. GILPIN, CLEVELAND, OHIO

(New York Meeting, February, 1916)

THE application of electric power for driving mine hoists handling heavy loads at high speeds has recently been extended by the installation of what is probably the largest electrically driven hoist in this country at the Granite Mountain shaft of the North Butte Mining Co., Butte, Mont. While the weight of rock which is hoisted by this machine is not the largest now being handled by electric mine hoists, the speed of hoisting is so, and the combination of the high speed with the heavy load has resulted in a machine of considerable interest to all those actively engaged in mining. Aside from mere consideration of size, this installation deserves attention because of the careful attention given, not only to the question of economy of operation, but to the details of construction and installation of the entire equipment.

The conditions to be met in this installation were as follows:

Weight of rock to be raised per trip, 7 gross tons	15,680 lb.
Weight of skip	8,000 lb.
Weight of cage	1,800 lb.
Maximum depth of hoisting	4,000 ft.
Diameter of rope	1⅝ in.
Total weight suspended on one rope from drum	42,000 lb.
Normal hoisting speed	2,700 ft. per minute.
Maximum hoisting speed	3,000 ft. per minute.

Desired capacity—200 tons per hour from 4,000 ft. depth.

The ore is mined in several levels and hoisting is carried on from the 800-ft. level down to the 2,700-ft., which is at present the maximum depth from which the ore is raised regularly. Development is proceeding and a depth of 2,900 ft. has already been reached. The rock is handled underground in cars holding ¾ ton, and about 10 cars are the usual load for a skip. All of the ore is dumped into underground bins, which are of different capacities varying up to 250 tons. From the bins the ore is fed into the skips without passing into measuring pockets. At the surface the

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ore is delivered into a series of storage bins, from which it is discharged by gravity into railroad cars and shipped to the smelters.

With a high cost of fuel and the cost of electrical power relatively low, the most economical hoist apparatus is a simple geared hoist driven by an alternating-current motor with drum control. Its simplicity, however, carries with it certain disadvantages which become more pronounced as speeds of hoisting are increased. At this mine, where the service requires a combination of high speeds and heavy loads, the first choice of equipment is a hoist direct connected to the motor with voltage control. In the many questions that enter into the determination of the best hoist-

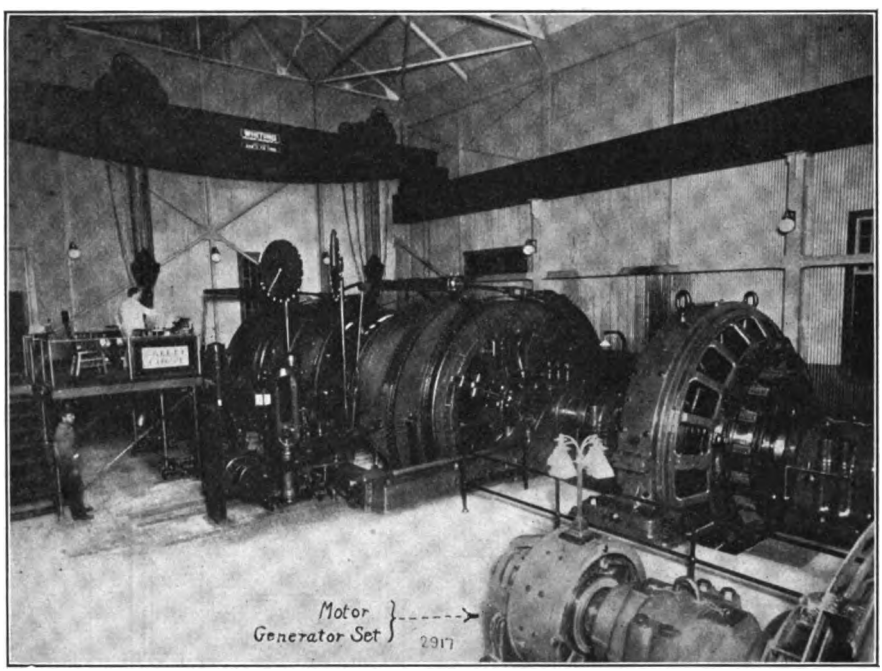

FIG. 1.—HOIST ROOM AT THE GRANITE SHAFT OF THE NORTH BUTTE MINING CO.

ing equipment for any given set of conditions, first cost may finally be pitted against sureness of control, freedom from interruption of service, and safety. The safety feature was given as careful attention as any of the other items entering into the consideration of the hoisting equipment to be installed.

In the present instance, a first-motion hoist with a motor-generator flywheel set was chosen as the most desirable equipment. Fig. 1 shows a part of the hoist room with the equipment installed, and Fig. 2 shows a plan of the hoist room. The hoist house is of steel, with steel sides and roof. An overhead crane and runway are provided, so that the heaviest parts of

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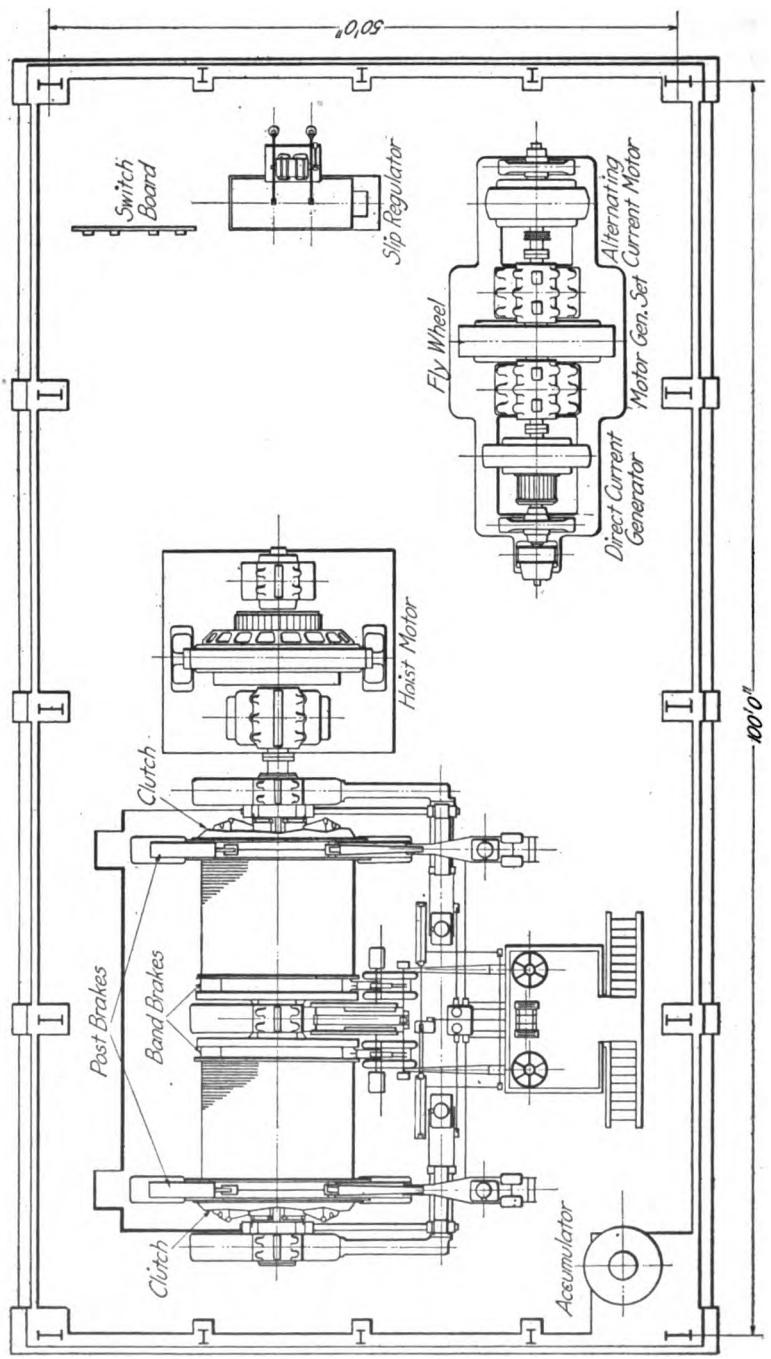

FIG. 2.—PLAN OF THE HOIST ROOM.

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the equipment can be handled in case repairs are necessary. The hoist is located 425 ft. from the head frame, the ropes between the head frame sheaves and the drums being supported on sheaves mounted on special towers.

The hoist consists of two 12-ft. drums fitted with a clutch, post brake and band brake. The drums are mounted on a shaft supported in three bearings, this shaft having a flanged coupling to connect to the motor. The clutches and post brakes are operated by oil cylinders, the pressure being supplied by an accumulator and an electrically operated pump. The band brakes are operated by hand wheels. All of the operating levers are grouped on a large elevated platform with double stairways. The control and reverse levers are separated, but so interlocked that when the control lever is in the "on" position the reverse lever cannot be moved. The safety devices include:

1. A mechanism for moving the control lever to the "off" position when the skip has reached a predetermined point, holding the lever in this position until the reverse lever has been moved to the opposite position, the operator being thereby prevented from starting in the wrong direction.

2. Two solenoids which automatically apply the post brake if the skip is carried too far after the current has been cut off. An indicator with large dial is provided for each drum; for accurately spotting the skip or cage the brake rings on the drums next to the middle bearing are extended 8 in., affording a large surface on which to paint marks.

On the platform in front of the operator are a panel, holding a voltmeter and an ammeter, and a target connected to the reverse lever showing which drum is hoisting. Grouped around the sides of the platform are the signals, gongs and lights.

The drums are 12 ft. in diameter by 9 ft. 4 in. face, grooved to hold 5,000 ft. of 1⅝-in. rope in two layers. The drum shells, brake rings and spiders are made of cast steel, the latter being fitted with heavy bronze bushings.

The clutches are designed to take a load of 50,000 lb. on a 12-ft. diameter, with a factor of safety on all parts of not less than 8. The clutches are of the flat friction type, consisting of two heavily ribbed annular rings faced with wood supported on a six-armed spider keyed to the shaft. These rings clamp a flat steel plate bolted to the drums, and are moved by six sets of toggle arms connected to a sliding sleeve and rock shaft operated by an oil cylinder. All of the parts of the clutches are made of steel. The clutches and motor were subjected to a load of 2½ times their rated capacity and developed no weakness.

The post brakes are made of plates and angles in the form of a box girder. They are of the parallel-acting type applied by weights and released by oil cylinders. The band brakes are for emergency service and are operated by hand wheels and screws, with provision for operation by

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power later on if desired. All of the brakes are lined with basswood blocks.

The bearings are of the pedestal type with quarter-boxes adjustable both vertically and horizontally. A continuous gravity-feed oiling system with tanks and filters is provided for lubrication. The drum shaft is of open-hearth forged steel, 22 in. diameter by 40 ft. 5½ in. long, with a flanged coupling forged on end. All of the operating connections and safety devices are placed on or above the floor level, in full sight, so that any derangements of any of the working parts can be quickly observed.

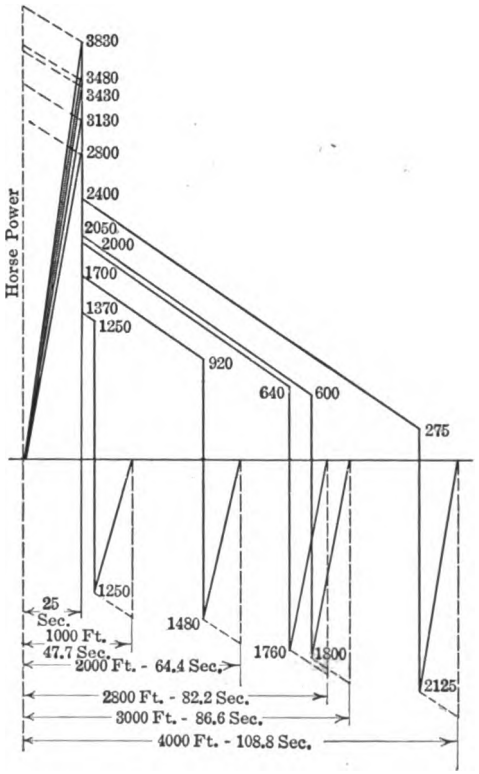

FIG. 3.—CALCULATED HOISTING SPEED AND POWER CURVES.

The combined weight of drum shaft with two drums and two clutches is 300,000 lb.; the radius of gyration is 4.86 ft.

The electrical equipment consists of a 1,850-hp., 71-r.p.m., 550-volt direct-current, direct-connected hoist motor, directly connected electrically to a generator of equivalent capacity which is driven by a 1,400-hp., 505-r.p.m., AC motor. A 100,000-lb. flywheel is mounted between the motor and the generator of the motor-generator set. The flywheel is turned smooth and is inclosed in a steel case which reduces not only the windage friction, but the noise; the latter often being a very annoying

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feature with a set of this kind. Both the hoist motor and the generator are designed to carry high overloads.

Forced lubrication is used on the motor-generator set, and the wheel bearings are water-cooled. The slip regulator is also water-cooled.

The field current of the generator is not regulated directly, but by means of magnetic switches operated by the master controller. The

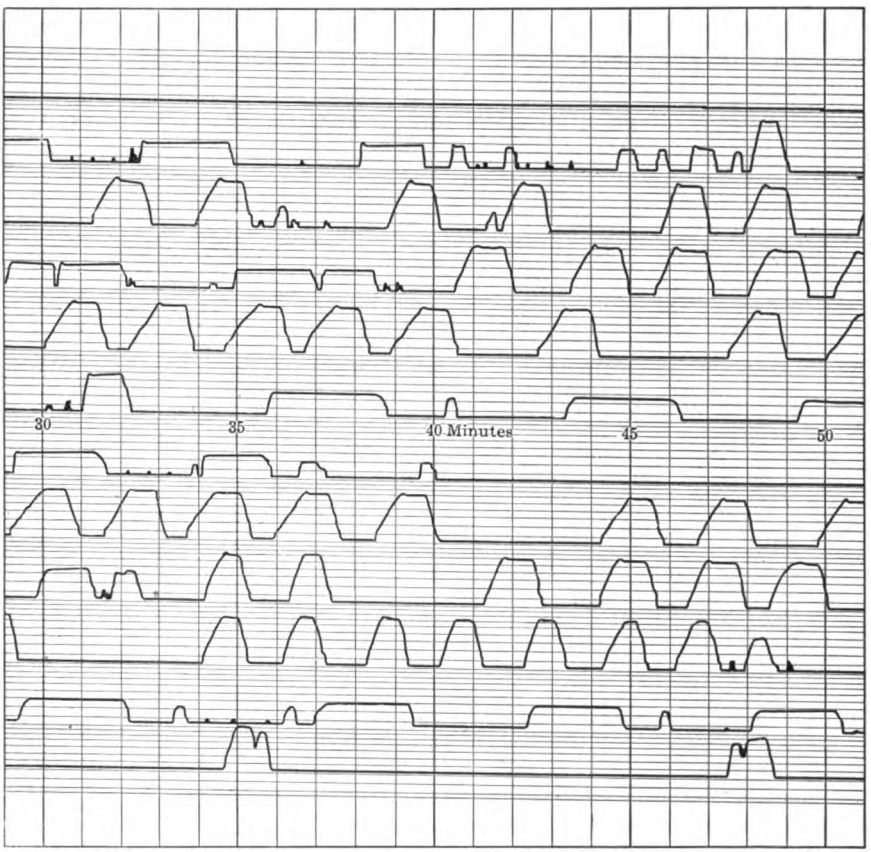

FIG. 4.—TACHOGRAPH RECORD SHOWING TYPICAL HOISTING SPEEDS AT THE GRANITE SHAFT.

wiring for the latter is all brought to a junction box on the operating platform, so that the operator may know if any changes are being made. All other wiring is carried in the basement, conduit work being used throughout with the exception of the large mains between the motor and generator which are mounted in porcelain cleats in a substantial manner. Every connection is plainly marked so that it can be checked up by reference to the wiring diagram.

The operation of the hoist is extremely easy, merely one air brake

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being commonly used; this air brake, in fact, is usually not applied until the rope speed is reduced to almost nothing. At present the hoist is not pushed to its capacity and consequently the operators accelerate and retard in a leisurely manner. Fig. 4 shows some typical speed records from the tachograph with which the hoist is provided. Fig. 5 is a power curve taken when the operator was hoisting at a speed about 10 per cent. below normal from a depth of approximately 2,250 ft. Fig. 3 shows a series of calculated curves. As a matter of interest, the values from the 1,000-ft. level curve and 3,000-ft. level curve have been applied to equation 4 in the note appended to this paper, with the result that a flywheel sufficient to smooth out all the peaks is indicated, based on assumed values for demand and meter charges. Calculations made from these curves

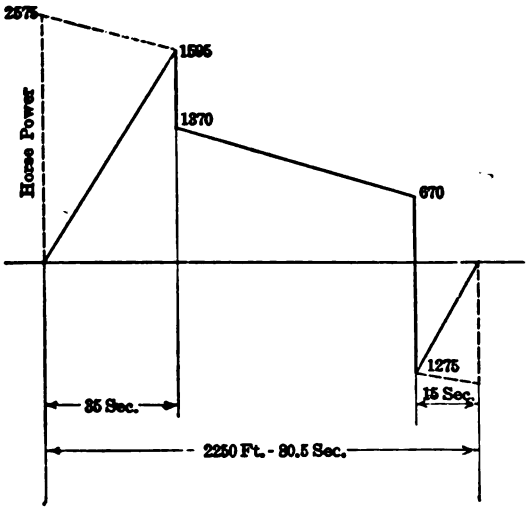

FIG. 5.—POWER CURVE WHEN HOISTING AT A SPEED 10 PER CENT. BELOW NORMAL FROM A DEPTH OF 2,250 FT.

indicate for the 1,000-ft. level that a 90,000-lb. wheel will be required, while for the 3,000-ft. level a 111,000-lb. wheel will be necessary, a minimum time of 15 sec. being allowed for loading in each case. These figures are approximate and are in no way related to those prepared by the manufacturers of the electrical equipment, who, of course, have access to more complete data on the characteristics of the electrical equipment. In these calculations, the slip was taken at 20 per cent.

The input curve at present reaches a value of 1,140 kw. and consists of a series of peaks, since the interval between trips is too great to give the constant power input that would be attained under maximum rates of operation. The switchboard instrument indicates a no-load loss for the flywheel and motor shunt field, etc., of approximately 90 kw. or about 120 hp.

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It should be borne in mind that the equipment was installed for conditions which will prevail in the future, as much as for those which now are attained.

Preliminary Calculation of Flywheel Motor-Generator Sets

The subject of motor-generator hoisting sets has been covered so many times that it is difficult to offer anything new. The writer has endeavored, however, to develop a few general principles which may be of use to those interested in this subject.

In the first place, this system is often considered for hoists where it has no proper application. Take for instance a hoist where the rope speed is comparatively low. Though the peak loads on such a hoist may indicate a saving in demand charge which appears attractive, it should be borne in mind that the efficiency of an Ilgner system is much lower than that of a straight induction motor, so that the demand charge will not be

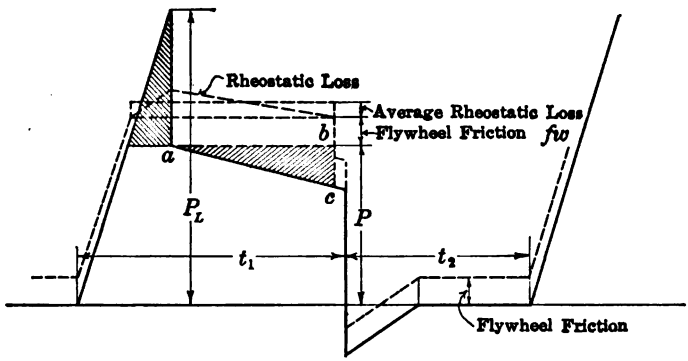

FIG. 6.—POWER CURVE USED IN CALCULATION OF FLYWHEELS FOR MOTOR-GENERATOR SETS.

reduced as much as would appear. Moreover, if the hoisting speeds do not require a motor-generator type of control, the substitution of an Ilgner system for an induction-motor drive will greatly raise the fixed charges, which will generally more than offset the saving in demand charges.

There is another class of hoists in which the splendid control of the motor-generator system is essential, but in which the peaks are not of sufficient magnitude to indicate the use of a flywheel. In cases of this kind, if the company supplying power will disregard peaks under 5 seconds, it does not pay to add the flywheel, since the acceleration load area is triangular and much of it will not carry any charge. As the load comes on gradually, there is little disturbance on the power lines, so that the power company can afford to make such a concession.

Finally, there are many instances in which a short inspection of the load cycle and the power contract will make it certain that a flywheel system is necessary. Just how much of the peak it is desirable to cut off is

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not so easily determined. The following method of calculation may, perhaps, be of interest.

A curve should be plotted, similar to the one shown in solid lines on Fig. 6, this curve to represent the power at the flywheel shaft. The peak cost per horsepower per trip (C_1) should next be determined by dividing the peak charge per horsepower per month by the probable number of trips per month, allowing for the losses in the alternating-current motor. The current charge (C_2) per horsepower-second should be figured.

It is obvious that the acceleration peak of the curve will be comparatively easy to do away with, and this point should be first considered. Assuming a constant maximum slip, s, for the flywheel (expressed as a decimal fraction of normal speed), and calling the weight of the flywheel, W, energy from the wheel (in horsepower-seconds) $= \frac{Wv^2(2s - s^2)}{2g \times 550}$ where v is the equivalent linear velocity of the flywheel in feet per second ($v = 2\pi$ rev. per sec. $\times$ radius of gyration of wheel).

This expression may be more conveniently written, nW; and to obliterate the acceleration peak it is only necessary to equate this value to the energy in the acceleration peak above the gravity load.

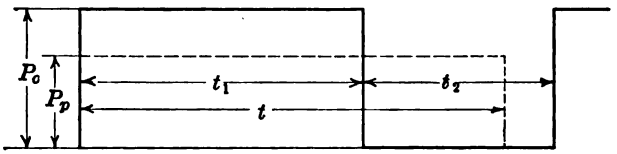

FIG. 7.—LOAD CURVE.

Referring to Fig. 6, re-acceleration would be represented by the area, *abc*. The actual peak at the wheel is P plus the wheel friction and the rheostatic losses, and the power curve has assumed an approximately rectangular shape. It would not, therefore, be far out of the way in figuring the desirability of a further increase in the flywheel, to assume that the load curve is a rectangle as shown in Fig. 7, in which t_1 is the time of hoisting up to retardation, P_o the average power at the wheel required by the hoist during this period, and t_2 the time of retardation plus the time of rest. P_p is the power delivered at the wheel shaft by the alternating-current motor, the difference between P_o and P_p being supplied by the wheel. The energy $P_o t_1$ then equals $P_p t$. The negative power has been disregarded, as it does not ordinarily enter largely into the final result.

Let f equal friction horsepower per pound of flywheel, and T = total time wheel runs per trip. Since approximately all the power passes through the motor-generator set while the latter is accelerating or retarding, the average rheostatic loss in power equals $\frac{s}{2}(P_p + fW)$. The peak

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load, therefore, equals $P_p + fW + \frac{s}{2}P_p + \frac{s}{2}fW = \left(1 + \frac{s}{2}\right)(P_p + fW)$

$P_p = P_o$ less the power supplied by the wheel $= P_o - \frac{nW}{t_1}$

Substituting,

$$\text{peak load} = \left(1 + \frac{s}{2}\right)\left(P_o - \frac{nW}{t_1} + fW\right)$$

Let the peak load cost of power per trip $= F_1$.

$$F_1 = C_1\left(1 + \frac{s}{2}\right)\left(P_o - \frac{nW}{t_1} + fW\right)$$

The rate of change of peak cost with regard to wheel weight is the first derivative of F_1, or $\frac{dF_1}{dW}$

$$\frac{dF_1}{dW} = C_1\left(1 + \frac{s}{2}\right)\left(f - \frac{n}{t_1}\right) \qquad (1)$$

With ordinary values for n, t_1 and f, this rate of change will be negative, showing a decrease in peak cost for an increase in wheel weight.

The energy consumption at the flywheel shaft is the sum of three quantities: the energy demanded by the hoist, the energy consumed by the flywheel, and the energy lost in the slip regulator rheostat. The first of these quantities is represented by $P_o t_1$, and the second by TfW, it being remembered that T is the total running time of the wheel per trip of the hoist.

The third quantity (E_3) is equal to the energy lost when introducing resistance plus the energy lost while resistance is being cut out; it may be expressed as follows:

$$E_3 = \frac{s}{2}(P_p + fW)t_1 + \frac{s}{2}(P_p + fW)(t - t_1) = \frac{s}{2}(P_p t + tfW) = \frac{s}{2}(P_o t_1 + tfW)$$

The quantity t is a variable dependent on W, and may be expressed in terms of the latter; but for approximate calculations of this kind, it is much simpler to equate t to T, which is its limit. The error so introduced, as will be shown hereafter, will be small, when determining the rate of change of energy cost.

The total energy, E, may therefore be approximated as follows:

$$E = P_o t_1 + Tfw + \frac{s}{2}(P_o t_1 + TfW)$$

Let the meter cost per trip be F_2. Then,

$$F_2 = C_2\left[P_o t_1 + TfW + \frac{s}{2}(P_o t_1 + TfW)\right]$$

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The rate of change of meter cost $= \frac{dF_2}{dW}$

$$\frac{dF_2}{dW} = C_2\left(Tf + \frac{s}{2}\,TF\right) = C_2\left(1 + \frac{s}{2}\right)Tf \qquad (2)$$

(It will easily be seen that if t equals 0 instead of T, the term $\frac{s}{2}$ will be eliminated; with 15 per cent. slip, $\frac{s}{2} = 0.075$. The error is therefore usually much less than 7½ per cent.)

$\frac{dF_2}{dW}$ is a positive expression and shows F_2 increasing with the weight of the wheel. If the derivative of F_1 is negative and is greater numerically than the derivative of F_2, the power cost per trip will decrease continuously as the wheel weight is increased, until the peaks are entirely smoothed out. If the derivative of F_2 is the greater, any increase in the wheel weight will increase the cost of power. This relation may be expressed as a ratio of the two derivatives

$$\frac{dF_1}{dW} \div \frac{dF_2}{dW} = \frac{dF_1}{dF_2} = \frac{C_1\left(1 + \frac{s}{2}\right)\left(f - \frac{n}{t_1}\right)}{C_2Tf\left(1 + \frac{s}{2}\right)} = \frac{C_1\left(f - \frac{n}{t_1}\right)}{C_2Tf} \qquad (3)$$

If e_1 equals the average efficiency of the alternating-current motor; R_1, the peak charge per horsepower per month; R_2, the meter charge per horsepower-hour; D, the working hours per month, and Z, the average trips per working hour, then

$$C_1 = \frac{R_1}{DZe_1};\ C_2 = \frac{R_2}{3{,}600e_1};\ T = \frac{3{,}600}{Z};$$

and

$$\frac{C_1}{C_2T} = \frac{R_1 3{,}600e_1Z}{DZe_1R_2 \times 3{,}600} = \frac{R_1}{DR_2}$$

Substituting in (3),

$$\frac{dF_1}{dF_2} = \frac{R_1}{R_2Df}\left(f - \frac{n}{t_1}\right) \qquad (4)$$

If the numerical value of $\frac{dF_1}{dF_2}$ exceeds unity and is negative, all the peaks should be removed; otherwise additional flywheel weight will only increase the cost of power.

It may be objected that a flywheel merely large enough to obliterate the acceleration peak will not in every case produce an approximately rectangular-shaped power curve. This objection is met by an inspection of the typical load cycles shown in Fig. 8. In both cases h will be the average power for the time $t_1 + t_2$. Let W_1 be the wheel required to reduce the triangular load to h, and W_2 be a similar wheel for the rec-

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tangular load. Without going into the mathematics on this point, the relation of the wheel weights is as follows:

$$\frac{W_1}{W_2} = \frac{(t_1 + 2t_2)^2}{4t_2(t_1 + t_2)}$$

When t_2 is very large as compared to t_1, W_2 will be as large as W_1; and since the reduction in peak for the rectangular curve is much less than for the other, if it is advantageous to reduce the rectangular peak, it will be still more so to smooth out the triangular curve. The other extreme is represented by the value of t_1 which is large compared to t_2; in actual practice, this ratio will hardly exceed 10 to 1 in cases where a flywheel is applicable. $\frac{W_1}{W_2}$ then becomes equal to $\frac{1.44\,t_1^2}{0.44\,t_1^2}$ or to 3.3, and $W_1 = 3.3W_2$.

The rectangular curve will show a reduction in peak of less than one-tenth, while the triangular curve will be reduced approximately one-half.

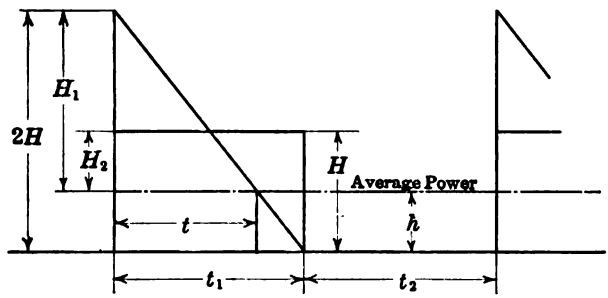

FIG. 8.—TYPICAL LOAD CYCLES.

Since the losses are roughly proportional to the wheel weights, the triangular load will show a ratio of saving over that of the rectangular load in the neighborhood of 5 to 3.3. It would appear, therefore, that the rectangular form of curve, for any ordinary conditions, is the limit of the other forms. Therefore, it may be assumed that if any hoisting cycle shows by means of equation (4) that its input should be made constant, a saving in power bills will result by so doing. The converse is not necessarily true; *i.e.*, a curve of markedly triangular form, which by the use of equation (4) shows a value of somewhat less than unity for $\frac{dF_1}{dF_2}$, should be further investigated by means of one or two trial wheel weights.

Referring again to the formula for $\frac{dF_1}{dF_2}$, approximate values may be placed on all the constants. The expression $\frac{R_1}{R_2}$ would probably lie between the ratios $\frac{50}{1}$ and $\frac{400}{1}$ with various power contracts; D (working hours

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per month) should approximate 600; f may be estimated roughly as 0.001 for ordinary speeds of flywheel (1 hp. per 1,000 lb. of wheel). Quantity n may be taken as 0.4, and the value of 0.15 for s would not be far off.

Substituting the greatest ratio of $\frac{R_1}{R_2}$ and the other values as stated,

$$\frac{dF_1}{dF_2} = \frac{400}{600 \times 0.001}\left(0.001 - \frac{0.4}{t_1}\right) = 666.7\left(0.001 - \frac{0.4}{t_1}\right) = 0.6667 - \frac{266.7}{t_1}.$$

Equating $\frac{dF_1}{dF_2}$ to -1, which is the ratio at which it will cease to be profitable to add flywheel, t_1 will equal 160 sec.

Substituting the minimum ratio of R_1 and R_2 $\left(\frac{50}{1}\right)$, t_1 becomes 31 sec. Therefore, when the time of hoisting, less the time of retardation, is less than 31 sec., and the working hours per month are about 600, it is extremely likely that a sufficient flywheel should be supplied to make the power input constant; if it is more than 160 sec., no flywheel should be added over that necessary to remove the acceleration peak. If 20 per cent. slip were assumed instead of 15 per cent., n would have a value of about 0.5, and the preceding time values would be increased about 25 per cent.

No attempt has been made to discuss the question as to the propriety of treating the slip, s, as a constant, but the writer believes that for most cases this value should be as great as the design of the motor-generator set will allow. Values of n can readily be worked out for various speeds of flywheels, but a discussion of the probable values of f, by those engaged in the design of this type of apparatus, would undoubtedly be of interest.

Discussion

K. A. Pauly, Schenectady, N. Y.—I read with interest Mr. Moeller's paper descriptive of the mechanical parts of the new electric hoist at the Granite Shaft of the North Butte Mining Co. and as the paper treats largely of the mechanical equipment I have little discussion to offer. There is, however, one question which I would like to ask and that is, why the discrepancy in hoisting cycles between the curves given in the paper read by Mr. Sykes for the American Institute of Electrical Engineers at San Francisco and those given in the paper under discussion? In Mr. Sykes' curves the acceleration period is shown to be 16 sec. for all depths from 2,000 to 4,000 ft. inclusive, while Mr. Moeller in Fig. 3 shows 25 sec. throughout the range and in Fig. 5 shows 35 sec. Mr. Moeller's curves also show the following reductions in the peak load during acceleration:

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Level, Feet	Mr. Sykes, Horsepower	Mr. Moeller, Horsepower	
2,000	3,500	3,130	Fig. 3
3,000	3,800	3,480	
4,000	4,100	3,800	
2,250		1,595	Fig. 5

As the purchaser's specifications called for 15 or 16 sec., I would like to know whether or not any modifications have been made in the requirements since the equipment has been installed.

The method of determining the most economical size of flywheel for a motor-generator set is ingenious and interesting, but we have found in our experience that approximate methods are of little value. In the first place the weight of the flywheel in most cases, if not all, is determined by the unbalanced cycle, and usually a wheel which will take care of hoisting even a reduced load, unbalanced, will be sufficient to equalize fully the balanced cycle. Our experience has also indicated that in almost every case where power is purchased on a peak-load basis it is more economical to equalize fully. Further, we have found it so easy to study the flywheel problem accurately that there is little excuse for an approximate method which only serves as an indication of a starting point for more accurate calculations, although possibly such an approximation might be of value to those less experienced in this work.

GRAHAM BRIGHT, E. Pittsburgh, Pa.—I made some of the original calculations on this hoist, and I would like to answer some of Mr. Pauly's questions. He says that Mr. Sykes' paper has shown the accelerating period to be 16 sec., while Mr. Moeller shows later 25 to 35 sec. That is due to two reasons, one is that the weights of the various parts are somewhat heavier than originally contemplated. The skips, cages, and rotating parts of the hoist weigh more than first figured on, while the average weight of ore is also a little more than first figured on. No one of the above individual parts is a large percentage of the total, but in the aggregate add considerably to the weight to be accelerated while getting the hoist in motion, and that, of course, means that either the power or the time must be increased.

The stand taken by the North Butte Mining Co. was that the ore could not be brought to the surface without pushing the hoist beyond its normal capacity, so a little less accelerating current than first figured on was used, and that, coupled with a heavier weight, simply lengthened the period, as indicated. Recently the cycle of operation has been changed by increasing the accelerating current to about 5,000 to 5,500 amp. The accelerating time is now from 18 to 24 sec. The actual heating during the entire cycle is less than it was before.

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Mr. Pauly's remarks concerning the flywheel, I think, possibly need a little explanation; I refer to his statement that where power is purchased on peak-load basis the flywheel is necessary. It all depends on how the peak is measured. For some time the power companies used instantaneous peaks for determining their demand charge, but that practice has largely disappeared. The time is lengthening out from 1 to 5 to 10 min., and even to 15 min. Many of the power companies are basing their charges on the integrated peak of from 5 to 15 min., so in none of our hoist cycles is there anything like that time; most of them were over in a minute or so, so that any peak load based on longer than that time should not require a flywheel. Unless the contract peak is based on a very short time, a flywheel should not be required for complete equalization. Sometimes the power companies' contracts are such that a light flywheel for partial equalization is advantageous although requiring a larger induction motor on the set.

FRANKLIN MOELLER, Cleveland, Ohio (communication to the Secretary*).—With reference to the remarks of Mr. Pauly on the discrepancy in hoisting cycles, I would say that the curves shown in this paper were based on observations made while hoisting was carried on under the ordinary operating conditions, and these curves have no connection either with those shown by Mr. Sykes in his paper before the American Institute of Electrical Engineers at San Francisco or with the purchaser's specification.

As to the method of determining the most economical size of flywheel for motor-generator sets, there was no intention on my part to offer any approximate method for doing so. The underlying idea of this part of the paper was to develop, if possible, a simple formula for determining whether, under a given set of conditions, the addition of a flywheel to the motor-generator set was justifiable or not.

* Received Mar. 29, 1916.

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Application of Electric Power to Mining Work in the Witwatersrand Area, South Africa

BY J. NORMAN BULKLEY, S. B., NEW YORK, N. Y.

(New York Meeting, February, 1916)

As electrical power is used to a greater extent on the Rand than in any other mining center, it is thought that a short description of the methods used and results obtained may be of interest. In comparing Rand practice with that on other fields, several general factors should be borne in mind: First, the mines are all working on large tonnages of low-grade ore (average $6.50). The annual crushing rate is about 27,000,000 tons. Second, the mines are all grouped under the financial and engineering control of mining houses, so that the results of a group, rather than an individual mine, are open for study, and practice follows more uniform lines. Third, coal of fair quality (about 12,000 B.t.u.) can be obtained for $2 to $3 per ton. Good supplies of condensing water are, however, scarce, and have to be carefully stored.

Power Supply

Power is supplied by the Victoria Falls & Transvaal Power Co., Ltd., and its subsidiary, the Rand Mines Power Supply Co. Both companies, which are practically one from an engineering point of view, operate the following stations, all feeding into the same system.

Station	Capacity Installed Turbo Alternators	Capacity Installed Steam-Turbine Driven Centrifugal Compressors
Brakpan	Two 12,500-kw. sets	
	Two 3,000-kw. sets	
Simmer Pan	Six 3,000-kw. sets	
Rosherville	Two 11,000-kw. sets	Six 3,500-kw. machines
	Five 9,600-kw. sets	Three 7,000-kw. machines
Vereeniging	Two 9,600-kw. sets	
	Two 12,000-kw. sets	
	162,200-kw. sets	

Total steam-turbine capacity installed, 204,200 kw.

There are also at the Robinson Central air station six motor-driven centrifugal compressors, each of 3,500 kw., equal to 22,000 cu. ft. per minute at 120 lb. capacity at 3,000 r.p.m. Power is supplied from these stations as 50-cycle, three-phase current generally at 40,000 volts,

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though the western end of the line is at present worked at 20,000 volts, and the Vereeniging tie line at 80,000 volts. The supply to the consumer is generally at 2,100 and 525 volts. The combined output of these stations is about 2,000,000 kw.-hr. daily, exclusive of air, with a peak load also exclusive of air of about 92,000 kw.

In addition to the supply furnished by the above companies, three groups furnish their own power, viz.:

	Station, Kilowatts
Randfontein Group	26,000
East Rand Proprietary Co	20,550
Kleinfontein Group	6,000

Price of Power

Standard contracts of the Victoria Falls & Transvaal Power Co. are for a period of not less than 12 years at 0.525 d. per kilowatt-hour as long as the monthly load factor is above 0.70, the load factor being based on the hour of maximum consumption. Provision is made for a periodical revision depending on cost of production and a division of profit with the consumers after certain deductions are made. As the other stations do not supply consumers outside their own group, no figures for them can be given.

Application of Electric Power

Reduction Works

(A) *Stamp Mills.*—In the modern mills of 2,000 to 1,900 lb. falling weight, the stamps are arranged five stamps on a camshaft, two camshafts being driven from a countershaft driven from one 50-hp. squirrel-cage motor running at from 500 to 600 r.p.m. This arrangement is generally the most convenient as it permits of overhanging both line and countershaft pulleys, so that belts can be easily removed, and the placing of motors on foundations at ground level, doing away with all motor platforms. It also makes a convenient arrangement when it is necessary to hang up stamps. When squirrel-cage motors are used, the absence of all brush gear permits the starting gear to be placed in front of the mortar box so that the mill man has a good view of the battery in starting up. With squirrel-cage motors there is no difficulty in starting up with stamps all down after a failure of power supply. Starting up can also be considerably eased by slacking off the belt tighteners. In some cases of conversion of old mills from steam to electric drive it was found impossible, owing to structural details, to adopt the plan outlined above; therefore a comparison of power required for such cases may be of interest.

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Mill *A* is of 75 1,250-lb. stamps all driven from one motor through line shaft; line shaft was in good alignment.

Mill *B* is of 120 1,250-lb. stamps weighted up to about 1,500 lb. falling weight and has two line shafts each with a motor driving 60 stamps; conditions of line shaft not so good as in Mill *A*.

Mill *C* is of 100 1,900-lb. stamps of which usually only 50 are running; drive is 10-stamp arrangement described above. Results are given in following table:

Mill	Monthly Tonnage	Kw.-Hr. per Ton Crushed	Number of Motors
A	14,910	10.42	1
B	28,260	8.13	2
C	33,454	4.84	10

(B) *Tube Mills.*—These are usually driven by belts direct from a 500 to 600 r.p.m. motor to tube-mill countershaft. Direct coupling to pinion shaft through flexible couplings with motor speed of 250 r.p.m. has been used in a few cases but without much success as the jar from the mill tends to break down the motor windings. On account of the high starting torque required, motors must be of the slip-ring type, and if the motor is properly proportioned to the mill with heavy starting resistances, there is no necessity for the employment of clutches or belt-shifting devices. The usual motor is 125 hp. operating at 600 r.p.m. for a 5 ft. 6 in. by 22-ft. tube mill; or 100 hp. for a 5 ft. 9 in. by 16-ft. mill.

(C) *Other Motors.*—The motors required for rock breakers, conveying plant and cyanide works are generally standard squirrel-cage motors belted directly to individual machines.

In the cyanide works a number of centrifugal pumps with direct-coupled motors were used, but owing to the necessity of adjusting speeds to suit heads, these have been discarded in favor of belt drive.

The number of motors required for a modern reduction works of about 40,000 tons monthly capacity will be roughly 125, and by intelligent selection the number of sizes required can be reduced to six or seven, so that spares will be a minimum.

The load factor of reduction works with 20.7 tons stamp duty can easily be kept about 86 per cent. and the average distribution for the newer works (40,000 tons per month capacity) will be as follows:

	Kw.-Hr. per Ton Milled
Stamp mill	4.29
Tube mill	6.00
Tailings wheels	0.87
Cyanide works	1.55
Breaking and sorting	0.68
Mechanical haulage	0.20
Lighting	0.23
Mill water supply pumps	0.65
Total for reduction works	14.45

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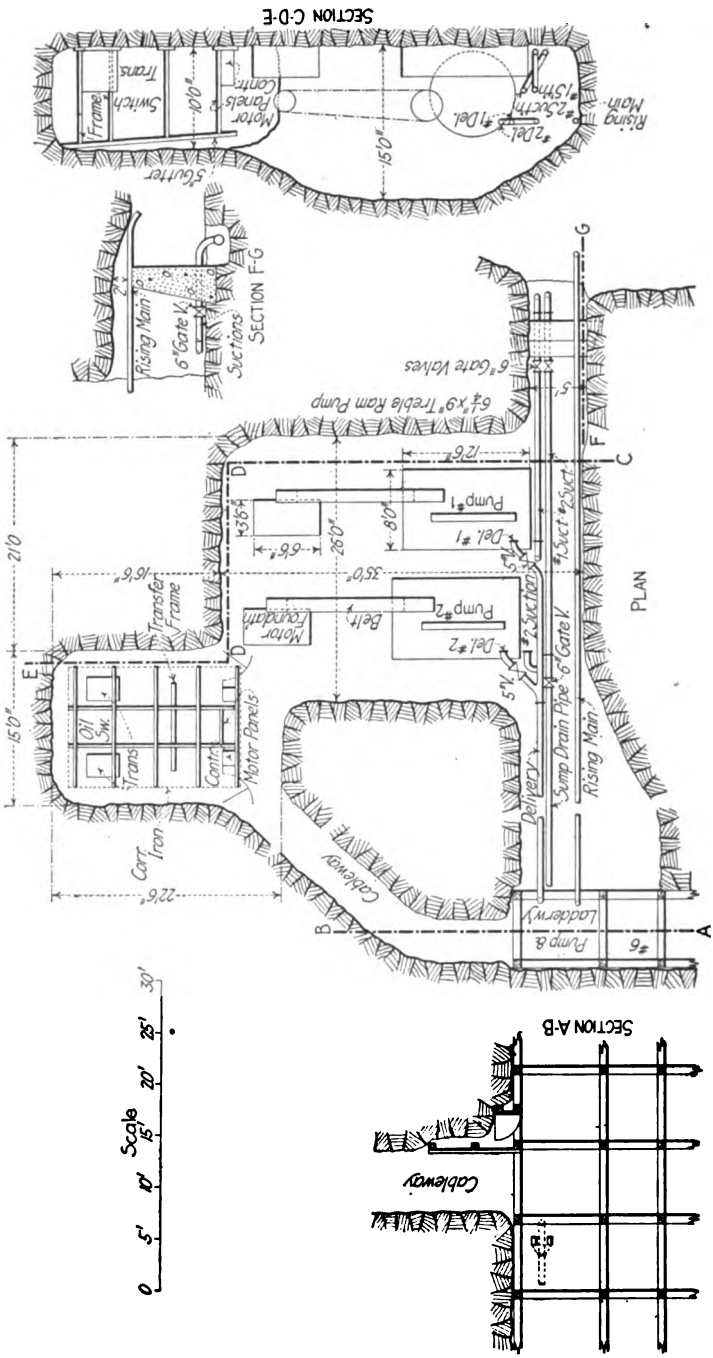

FIG. 1.—ARRANGEMENT OF PUMP STATIONS FOR 1,000-FT. LIFT, CINDERELLA CONSOLIDATED GOLD MINES, LTD

Pumping

Fortunately the majority of the Witwatersrand mines are comparatively dry, 10,000 gal. per hour units in duplicate usually being ample to handle all the water. In some sections, however, there are heavy flows, the East Rand Proprietary mines requiring pumps of 60,000 gal. per hour capacity. Practically all of the pumping plant is electrically driven, there being left in service few if any pumps of the Cornish type. Up to recent years all of the pumps have been of the triplex single-acting plunger type about equally divided between the horizontal and vertical patterns. Recently, however, speeds have been increased up to 60 r.p.m. and more horizontal pumps are used. Heads run from 1,000 to 1,700 ft. Most of these pumps are belted, not geared, to the motors, as experience has shown that the belt drive has lower maintenance cost. The depth of the shafts and amounts of power required makes the cost of shaft cables prohibitive at 500 volts, so high-tension cables are usually taken down the shafts. As high-tension motors, particularly those of moderate size, are unsuited for use underground, step-down transformers are required. Generally a three-phase transformer with oil switch on high-tension side is provided for each motor so no secondary switches are required. These transformers with their switches, etc., are placed in a fenced-off chamber to prevent any danger from the high-tension circuits, only the switch-operating handle and secondary circuits coming outside. A typical arrangement of such a pump station is shown on Fig. 1. The high-tension wiring in the transformer station is all bare copper carried on standard high-tension line insulators, the whole, together with necessary current transformers, switches, etc., being carried on an angle-iron framework so that all wiring can be fitted above ground. This particular station was laid out for 6,600 volts working and motors operated at 110 volts.

Centrifugal Pumps.—The advent of the high-lift centrifugal pump, notwithstanding its lower efficiency, was particularly attractive to engineers on account of its lower first cost and smaller space requirements. Several centrifugal pumping plants were therefore installed. These early plants, however, proved a source of trouble as the necessity of clearing the water of even traces of grit was not fully realized and the settling sumps provided were too small for their work. The experience gained showed what had to be done. The Durban Roodepoort Deep Gold Mining Co. installed a centrifugal pumping plant consisting of two units each handling 375 Imperial gallons per minute against a total head of 2,490 ft. For clarifying the water, rim launders were used to take off the water from the settling to the suction sumps. Each of these pumps is fitted with a 550-hp., 1,500-r.p.m. motor. The East Rand Proprietary Mines also have in service eight

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centrifugal pumps of 1,000 Imperial gallons per minute against 1,150 ft. head each equipped with a 550-hp., 1,500-r.p.m. motor. So far as the writer knows, both these plants are proving satisfactory.

Air Compressing

Air for the central mines of the Rand Mines is supplied by the power company from steam-driven compressors at Rosherville and motor-driven machines at the Robinson Central station. These stations supply about 2,250,000 tons of compressed air per annum. The machines are fully described in other papers.

The power company does not supply air to any other than the Rand Mines group. Owing to the bad influence of the compressor load on the load factor when the mine is operated, as is usually done on single-shift, the engineers of the remaining group have preferred, where steam plant was not entirely discarded, to keep the compressor on steam. The East Rand group operates a steam-driven, central air plant with reciprocating compressors and the Randfontein group operates a similar plant but with motor-driven units supplied with power from its own central station. There are a number of motor-driven compressors in use where there is no steam plant available. All these are of the reciprocating type, as the size of the largest unit, 7,500 cu. ft. per minute free air to 100 lb., does not allow the construction of an efficient turbine machine, the limit of which is about 10,000 cu. ft. free air. Nearly all are direct connected to either synchronous or induction motors, very few being belt driven. Two general types are in use, the vertical high speed and the horizontal low speed, both being compounded. Practically all of the later machines are fitted with some form of plate air valves automatically operated, as these have been found to give better results in every respect than any of the mechanically operated valves.

Governing.—Up to the present no satisfactory method of governing by means of a variable-speed motor has been developed; so resort must be had to some mechanical means with a constant-speed motor. The two methods commonly used are: (1) Throttling the intake; (2) opening an auxiliary governing valve for a portion of the stroke.

The first method is, of course, very simple, but throws the entire load on and off the compressor as the throttle opens and closes. Also, the efficiency is not as good as with the second method.

With the second method, by properly arranging the valve gear on the auxiliary valve, the compressor may be run at any desired fraction of the capacity with fairly good efficiencies on light load. Both methods work very well in practice.

Air Pressure.—The supply from the stations is at 120 lb. at the station and is calculated to be 100 lb. at the mine column. The usual

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pressure of the individual mine plants is 80 lb. but there is a decided tendency toward higher pressures.

Meters.—The use of purchased air necessitated the installation of air meters. With the accurate records of air consumed, study showed startling variations in the amount of air used per rock-drill shift on the various mines, and further study led to the development of systems for the regular inspection and repair of all drills, inspection of pipe lines and air pressures underground, all of which when summed up had the effect of reducing the air consumption and drilling costs by large amounts.

Hoisting

There are in operation 143 electrically driven hoisting engines, exclusive of winches. The combined continuous rating of these hoists amounts to over 74,000 hp., with an average of 517 hp. Hoisting engines may conveniently be considered under three classes: (*a*) winches; (*b*) sinking engines; (*c*) main hoists.

Winches

These are usually small engines with drums 42 in. diameter by 30 in. face, and a duty rarely exceeding 4,000 lb. rope pull at 500 ft. per minute. They are generally of the simplest construction and operated by a standard slip-ring motor through double-reduction gearing. Control is by means of a tramway-type reversing controller with metal resistances, the whole being mounted on the hoist frame so as to be readily portable. Such winches are rapidly displacing the air winches where power is available underground. In many cases, however, owing to the temporary nature of their use it does not pay to install special cables, and as air is required for drilling, the air winches are continued in service in spite of their inefficiency. In one case which came under the writer's notice, the question of increasing the compressor capacity arose. Investigation of the use of air showed a double 7 in. by 10-in. air winch in service. This was replaced with a motor-driven winch costing not over $1.50 per month for power, with the result that it was possible to put six additional drills on the compressor, and no increase in plant was required.

Sinking Engines

The high speed and great size of the main engines required on the deep shafts of the Rand precludes their being used during the sinking period and it is customary to install for this work special temporary engines which are usually placed in front of the positions to be occupied later by the main engines.

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Sinking engines must be capable not only of handling the buckets or skips but also of swinging all shaft timbers, etc., into place easily and safely without the necessity of reslinging them on chain blocks or tackles. Owing to difficulties in control, this requirement is not easily met with steam winders, though it presents no difficulties with electric drive. Furthermore, as the work of these hoists finishes with the sinking period, capital expenditure must be kept as low as possible. Consideration of these points has led to the development of the following type of sinking engine which has proved extremely satisfactory in service.

Engines are double drums 8 ft. diameter by 2 ft. 3 in. face. Each drum with its brake ring is keyed fast to an independent shaft with two bearings. These shafts also have keyed to them a main gear for each drum. The pinion shaft is common to both drums and carries the two loose pinions, which are clutched to the shaft by sliding-jaw hand-operated clutches. The brakes are of the post type, hand-operated through rack and pinion gear with hand wheel. A foot-controlled band operating on a brake disc on pinion shaft is also provided for maneuvering. The motor is of standard two-bearing slip-ring induction type of about 500 r.p.m., connected to the pinion shaft through machine-cut herringbone gears and flexible coupling. Control is by means of a liquid rheostat with pump and movable weir, reversing being accomplished by means of either contactors or oil switches. The usual load on these engines is 3,000 lb. net rock at a hoisting speed of 1,000 ft. per minute, though in laying out a new sinking plant the writer would be inclined to raise this speed to say 1,500 ft. The plant for the 3,000 ft. Central Shaft of the Cinderella Consolidated Gold Mining Co. was laid out before satisfactory delayed-action electric blasting fuse was available. Since this was the first deep shaft in the Transvaal to be sunk with electric sinking engines, it was deemed advisable to provide a sure means of hoisting the sinkers away from the blast after the round had been lit, and thus prevent accidents which might arise through failure of power supply.

This was accomplished by providing a synchronous motor fitted with necessary starting motor, an exciter wound for double normal voltage and an 8-ton 10-ft. diameter flywheel running at 750 r.p.m. After the blasting signal had been given, this set was started up in the ordinary way and connected to the line through a reverse power relay; the sinkers were not allowed to light up until they had received a signal that this had been done. Then in case of power failure the reverse power relay operated, leaving the synchronous machine driven by the flywheel operating as generator disconnected from the line but connected to the hoist so that the bucket could be pulled away. Actually, however, as it was necessary to stop the set after the blast, the reverse relay was tripped by hand every blast, and the sinkers hoisted by means of the flywheel. The records show that through power failure this set was required three times

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in two years' work. The operation of the entire plant was most satisfactory and the control was so effective that the general feeling of the underground staff was strongly in favor of electrical operation as against steam.

Sinking Incline Shafts

With these shafts the conditions differ, for sinking is generally carried on through a considerable portion of the life of the mine and the whole of the available space in the shaft above the lower levels is usually required to carry on the necessary work of the mine, so that it is hardly possible to provide a special sinking engine on the surface. If the main

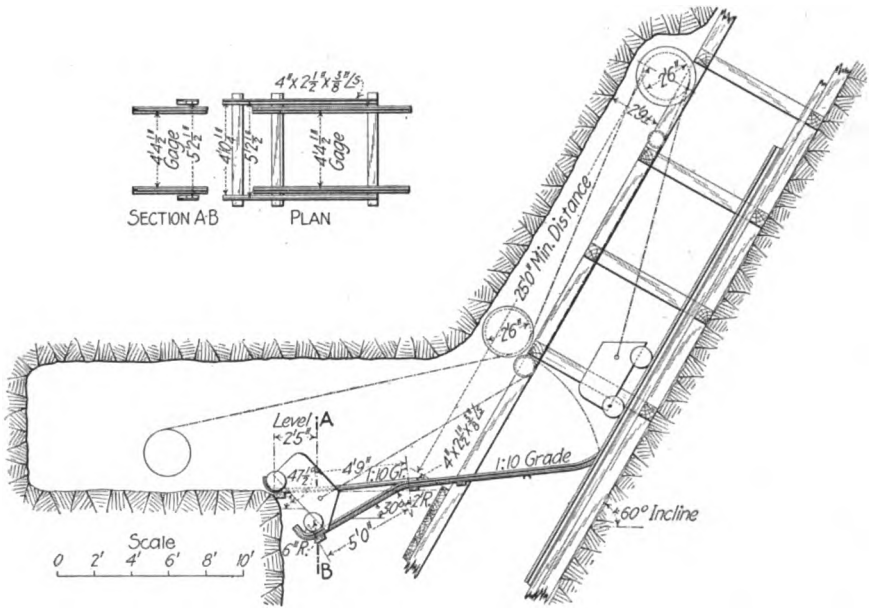

FIG. 2.—1¼-TON SINKING SKIP AND OUTLINE ARRANGEMENT OF TIP, KIMBERLEY SHAFT, ROODEPOORT UNITED MAIN REEF GOLD MINING COMPANY, LTD.

engines are used, the big skips are apt to give trouble on the temporary tracks at the shaft bottom as well as being difficult to load by shoveling. It is also usual to have only one skipway carried down to the bottom, necessitating unbalanced hoisting. Where electric hoists are used, unbalanced hoisting is troublesome because the motor is usually proportioned to give its best economy on balanced load and will overheat on continued use with unbalanced full load requiring reduced skip loads. All these considerations require the use of some special arrangement for sinking incline shafts. One of the most common schemes is the use of a small winch installed in the bottom level crosscut and hauling to the main skip box, the whole arrangement being moved down when the station of

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the next level is reached. The frequent moving requires that the winch must be self-contained, portable, and that the head sheaves, tip, etc., may be cheaply erected and with a minimum amount of excavation that will not be utilized in the permanent station. Such an arrangement is illustrated in Fig. 2. With this scheme the only extra excavation is the V-shaped cut, about 20 ft. long by 2 ft. 9 in. high, above the center of the skip way, to clear the sheaves and rope. The tipping gear consists of a pair of ordinary drop rails and four pieces of bent rail, together with some timber, so that the whole is cheap and readily portable. This arrangement needs no further description; it has proved extremely satisfactory in service.

Main Hoists

The enormous capacity of the main power supply stations permits extremely large amounts of power to be thrown on and off without affecting the stations. Consequently it is unnecessary to consider the installation of any flywheel sets to equalize the power demands on the line, so that the choice of rheostatic or Ward-Leonard systems of control is uninfluenced by any considerations in regard to the stations, and the choice can be made entirely on the conditions governing the particular mine. Under these circumstances the choice of system would depend largely upon the required speed of hoisting and whether hoisting when once started can be carried on steadily without intermission. The question of capital cost must also be considered as the Ward-Leonard system must always be higher in first cost.

In regard to speeds, rheostatic control is unsafe for speeds beyond those which can safely be controlled (even when negative torque is exerted) by means of mechanical brakes. Counter-current braking cannot be relied on as alternating-current readings are not definite indications of torque. This speed limit for mechanical braking, the writer places at about 1,500 ft. per minute. It is true that in an attempt to overcome this difficulty several high-speed rheostatic control plants have been installed with eddy-current brakes, and these plants have worked fairly well, but the addition of the necessary extra parts, including storage-battery motor generator, etc., for operating the brakes, makes the cost nearly or as high as for a Ward-Leonard set and loses some of the incidental advantages of the latter. However, where the required capacity can be obtained with speeds not exceeding the limit set above, rheostatic control will give extremely satisfactory service. As the low-speed alternating-current motor required for direct coupling to the drum shaft is necessarily a machine of large diameter with little length along the magnetic iron, the frame is generally weak mechanically. This in connection with the poor efficiency and power factor of these slow-speed machines makes direct coupling undesirable except for

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machines of the largest size; therefore the majority of rheostatic hoists are single-reduction geared with helical machine-cut gears. On the Rand, where it was necessary to convert a number of existing steam hoists to electric drive, the gear-shaft bearings were placed on the motor base so as to make a self-contained unit and the gear shaft coupled to the existing crank shaft through a flexible coupling, an arrangement which gave minimum change.

Hoist Control.—Control for rheostatic hoists was by means of liquid rheostat with movable weir for torque control and oil switches or air-brake contactors for direction control, both of these controls being operated from one lever. The oil switches on the smaller hoists were mechanically operated. On the larger machines compressed air and solenoids were used. However, experience proved the use of compressed-air operation unsatisfactory. Contactors were, of course, solenoid operated.

Ward-Leonard Hoists.—While the addition of the motor-generator set and the necessary control wiring make this system seem very complicated in written description, in reality after it is once installed it gives no more trouble than rheostatic control. As it is possible to use dynamic electrical braking with direct-current motors no speed limits are imposed by considerations of safety in the hoist itself, speed being governed by the permissible hoisting speed in the shaft. In this case the mechanical brakes are required for little more than locking devices at the end of the trip. This was clearly brought out at the Cinderella Consolidated shaft (4,040 ft. in depth) where after two years' service the tool marks were not worn off the brake path, the mechanical brakes not having been applied until the last 6 in. of drum travel. Owing to the high speeds usually employed with Leonard hoists, the motors are of fairly large size so that a good motor for direct coupling to the drum shaft can be obtained and thus avoid commutator troubles which might arise through pounding of the gears. The hoist motors are provided with interpoles and sometimes compensating pole face windings, with a result that sparkless commutation is secured and commutator troubles are practically unknown. Speeds of the motor-generators are only limited by considerations of mechanical and electrical design and can be fairly high, thus insuring an efficient machine at reasonable cost. These sets are usually provided with both interpoles and compensating windings on the generator. In some cases the Deri distributed shunt winding is used with marked success. The writer has installed motor-generator sets for hoist service that after three years of continuous service showed polished commutators which had never been turned down and which were in better condition than on the day they were started.

Control.—This is by means of the well-known regulation of the generator field and hardly needs comment. There are systems of control using rotary converters, and others with what might be termed, "buck-

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ing motors" but, so far as the writer knows, these have not been used on the Rand. When considering Ward-Leonard control, one fact should be borne in mind and that is that every position of the control lever is equivalent to fixed generator excitation and as the hoist motor has shunt characteristics the hoist speed must be practically definite, no matter what the load is or even if the torque be negative. This, of course, makes the control very easy for the driver.

Results with Electric Winding

In a paper of this sort it is practically impossible to present tabulated comparisons of costs with steam and electric winding, as in most cases the steam winders were supplied from boiler plants in common with other

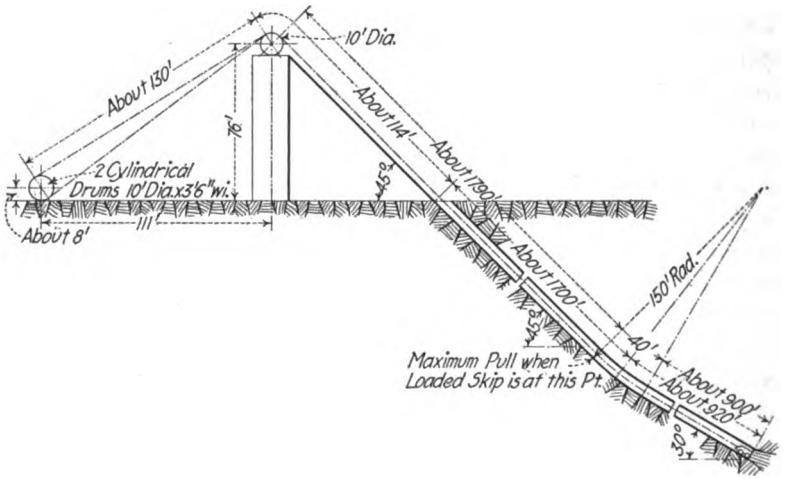

Fig. 3.—Rheostatic Winder 5-Ton Hoist No. 2 Shaft, Van Ryn Gold Mining Estate, Ltd.

engines so the amount and cost of steam used by the winders was a matter of doubt. Careful comparisons showed the costs to be in favor of electric winding, the saving in the case of some of the deep shafts (4,000 ft.) amounting to at least 12½ c. per ton. One unlooked for result was a marked decrease in the cost of maintenance of shaft guides, owing to the steadier turning moment of the electric winder. As to safety and reliability there can be no question but that under Rand conditions the advantage is strongly with the electric hoist. As a comparison of the efficiencies of Ward-Leonard and rheostatic control, two shafts having conditions as nearly alike as possible have been selected. These are No. 2 of the Van Ryn Gold Mining Co. (shown in Fig. 3) and that of the Meyer & Charlton Gold Mining Co. (shown in Fig. 4). The Van Ryn is equipped with a geared three-phase rheostatic winder

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handling 5-ton rock loads at 1,500 ft. per minute, with a monthly tonnage of about 22,000 tons. The Meyer & Charlton Gold Mining Co. has a direct-coupled Ward-Leonard hoist operating at 2,500 ft. per minute, with 5-ton skips of design and weight similar to those at the Van Ryn, but with a monthly tonnage of about 17,000 tons. Careful records of the performance of these hoists were kept on the forms shown in Figs. 5 and 6, and the results for a year's work plotted together with the shaft horsepower-hours (Fig. 7). A study of this curve will show that where the Ward-Leonard set has been kept working steadily, its efficiency exceeds that of the rheostatic set by about 5 per cent., but where the shaft horsepower-hours have dropped, the efficiency of the Ward-Leonard set has dropped with it, thus showing clearly the bad effect of intermittent

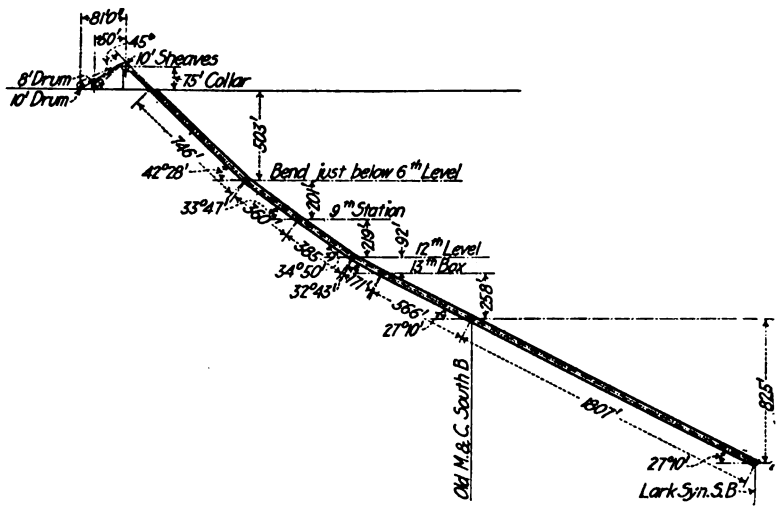

FIG. 4.—WARD-LEONARD WINDER, SECTION MAIN INCLINE SHAFT. MEYER & CHARLTON GOLD MINING COMPANY, LTD.

hoisting due to the practically constant losses of the motor-generator set, while with rheostatic control the only loss when the hoist is standing is that due to the controller pump motor which is negligible. The regularity with which hoisting can be carried on is well shown by the accompanying recording wattmeter and tachograph charts (Figs. 8 and 9) taken from the Meyer & Charlton plant. The wattmeter chart also shows clearly the effect of the dynamic braking in returning power to the line. Of course, to obtain such regularity it is necessary to provide suitable loading arrangements. For vertical shafts a modified form of the well-known Kimberley measuring loading chute has been found to answer well. For incline shafts, owing to the difficulty of quickly loading the skip full it was necessary to develop a form of measuring chute with a drop lip to deliver the ore well into the skip, and which in the closed position would

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GENERAL MINING AND FINANCE CORPORATION, LIMITED.

Mine Meyer & Charlton — ENGINEERING DEPARTMENT. — Month April 1915

RECORD OF WORK DONE BY 5 Ton W.L. — WINDER AT SHAFT Main Incline

Level.	Vertical Depth from Tip to Loading Bin. (Feet.)	Length of Rope from Tip to Loading Bin. (Feet.)	Rock Service. Number of Skips.	Rock Service. Tonnage.	Men Service. Number of Trips.	Men Service. Tonnage done in Raising Men.*	Material Service. Number of Trips.	Material Service. Tonnage Raised.	Number of Unbalanced Trips.	Tonnage due to Unbalanced Raising of Skip and Rope.	Total Tonnage Raised.	Useful Work Done in Raising of Loads in Shaft, H.P. hrs.†
1	2	3	4	5	6	7	8	9	10	11	12	13
4	396	690	123	534							534	213
5	481	718	51	221							221	107
9	796	1254	65	282							282	226
13	1088	1803	45	195							195	214
14	1162	1950	222	963	48	65					1028	1206
15	1264	2162	278	1207	19	14					1221	1546
16	1337	2344	13	56							56	75
17	1421	2527	231	1003	7	5					1008	1446
18	1518	2743	957	4153	180	344					4497	6598
19	1620	2958	1030	4470	196	402					4872	7975
20	1763	3272	494	2144	49	74					2218	3949
21	1914	3602	152	660	39	30					690	1334
											16822	25189
									25189 ÷ 47880 = .526			
TOTAL ...			3,661	15988	538	934					16,822	25,189

*Weight: 150 lbs. per Person. † Shaft H.P. hrs. = Total Tonnage × Vertical depth in feet / 990. Power Consumption 47880 K.W. hrs

FIG. 5.—PERFORMANCE RECORD OF WARD-LEONARD WINDER, MEYER & CHARLTON GOLD MINING CO., LTD.

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GENERAL MINING AND FINANCE CORPORATION, LIMITED.

Mine Van Ryn — **ENGINEERING DEPARTMENT.** — Month April 1915

RECORD OF WORK DONE BY 5 Ton 3 Dk. WINDER AT SHAFT No 2

Level.	Vertical Depth from Tip to Loading Bin. (Feet.)	Length of Rope from Tip to Loading Bin. (Feet.)	Rock Service. Number of Skips.	Rock Service. Tonnage.	Men Service. Number of Trips.	Men Service. Tonnage done in Raising Men.*	Material Service. Number of Trips.	Material Service. Tonnage Raised.	Number of Unbalanced Trips.	Tonnage due to Unbalanced Raising of Skip and Rope.	Total Tonnage Raised.	Useful Work Done in Raising of Loads in Shaft, H.P. hrs.†
1	2	3	4	5	6	7	8	9	10	11	12	13
2	339	490	41	180	128	38					218	75
3	437	625	58	255	394	118					373	165
4	540	—	—	—	173	52					52	28
5	695	1000	93	409	96	29					438	307
6	780	1120	380	1672	309	93					1765	1390
7	916	1313	545	2398	734	220					2618	2422
8	1053	1510	352	1549	240	72					1621	1724
9	1186	1700	450	1980	388	116					2096	2511
10	1303	1880	939	4132	438	131					4263	5611
11	1408	2060	272	5997	554	166					6763	8196
12	1583	2405	335	1474	260	78					1552	2491
Collar Bot.	63.		197	887							887	56
								Effy. 24976 / 40765×1.34			45.8%	
TOTAL			4662	20533	3714	1,113					21,646.	24,976.

*Weight: 150 lbs. per Person — † Shaft H.P hrs. = Total Tonnage × Vertical depth in Feet / 250 — Power Consumption 40765 K.W. hrs

FIG. 6.—PERFORMANCE RECORD 5-TON HOIST, NO. 2 SHAFT, VAN RYN GOLD MINING ESTATE, LTD.

J. NORMAN BULKLEY

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keep this lip well clear of the skip way. Such a chute is shown in Fig. 10 and has answered very well.

Comparative Cost of Electric and Steam Power

The writer would have liked to present tabulated statements of comparative costs of steam and electric driving of complete mining plants.

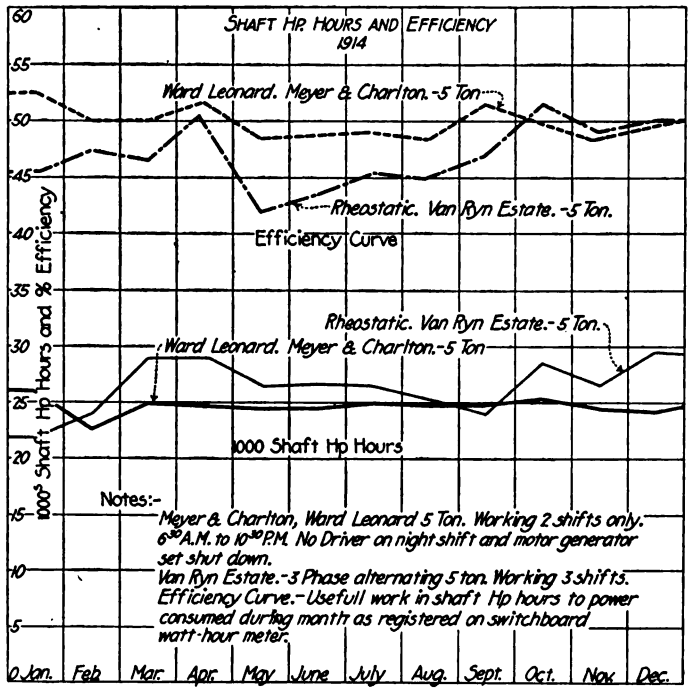

Fig. 7.—Comparative Efficiency Curve, Ward-Leonard Winder at Meyer & Charlton, and Rheostatic at Van Ryn Estate.

This was impossible as changes made in plants at the time of electrification would have made such comparisons seem unreliable except to those who had full knowledge of such changes. Also, in many cases steam had never been used. In one mine under the writer's charge, a complete change from steam drive to electric was made, the only boiler left in the plant being that supplying steam for cooking and wash water. In this particular case the question of the change was largely influenced by the fact that the mine was nearly at the end of its life. The acquisition of new ground gave the mine a much longer lease of life, and to continue with the existing steam plant would have meant a large amount of expenditure on power plant for renewals and repairs. For these reasons this installation could be regarded more or less as the equipment of a new mine. At the time this change was made the milling capacity was

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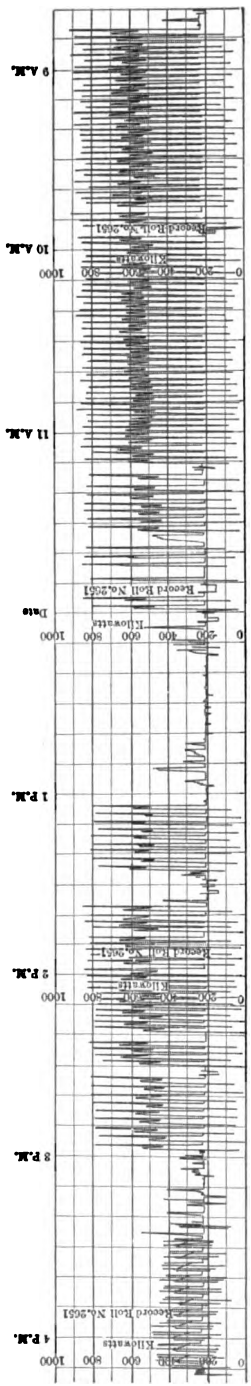

FIG. 8.—SECTION OF WATTMETER CHART, MEYER & CHARLTON PLANT.
Note.—Zero line has been raised to 200 to show braking power.

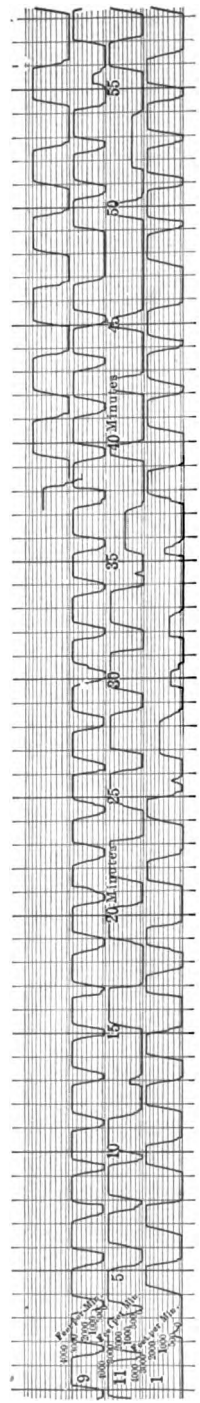

FIG. 9.—SECTION OF TACHOGRAPH CHART, MEYER & CHARLTON PLANT.

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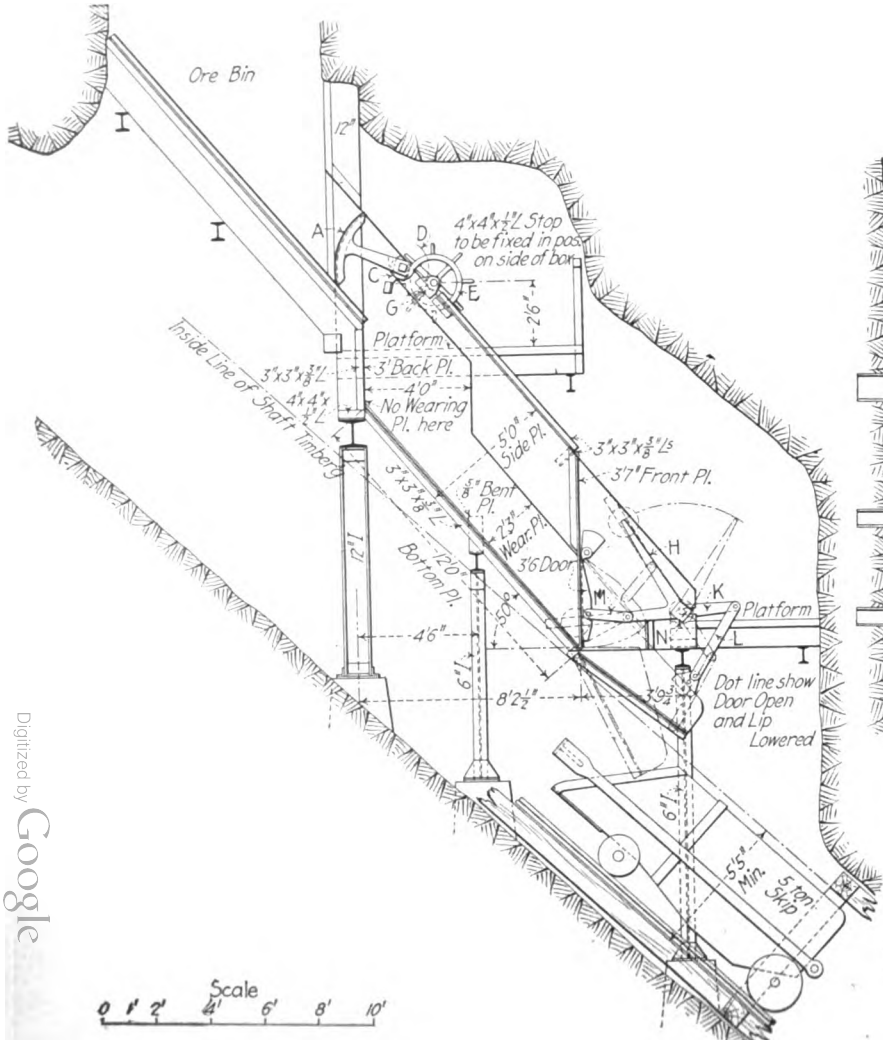

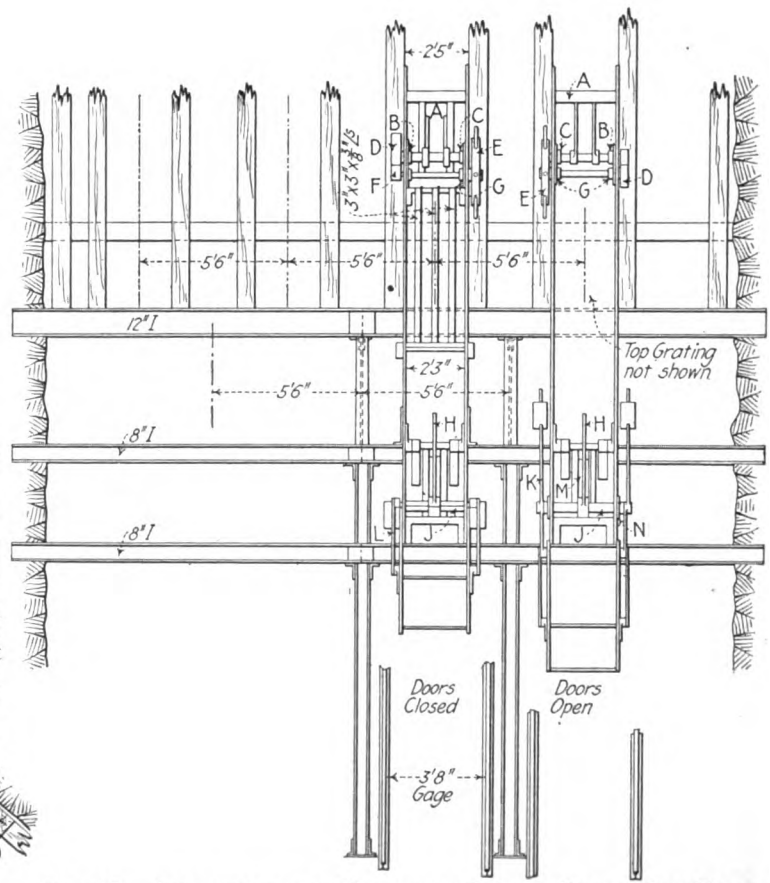

FIG. 10.—ARRANGEMENT OF MEASURING BOX AND CHUTE, DEEP SHAFT, MAIN INCLINE, CINDERELLA CONSOLIDATED GOLD MINES, LTD.

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increased from 15,000 to 17,000 tons per month and a new compressor having 33 per cent. more capacity installed. A study of the steam power costs for the last two years the plant was in operation and electric power costs for a year's service showed the saving in favor of electric drive to be $42,000 per annum without making any allowance for the increased power required by the additional tonnage treated and additional air supplied.

My thanks are due to E. Farrar, my former colleague in the General Mining and Finance Corporation, for his kindness in supplying the drawings and much of the information used in the paper.

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For those desiring further information on the subject of the paper the following bibliography has been prepared:

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Discussion

Graham Bright, E. Pittsburgh, Pa.—On p. 147 mention is made of a change from steam hoist to electric hoist. The arrangement mentioned, of using a geared motor, and having the gearing and countershaft in a separate, self-contained unit, and then connecting that to the hoist by means of a flexible coupling, is very good. Sometimes, however, it is necessary to change a geared hoist over to an electric hoist. In that case the motor could be either direct-connected to the countershaft or geared to the countershaft, making a double-reduction gearing. Trouble has been experienced with this method on account of the hoist not being designed for electric drive. It may be of a rather light construction which will not stand the forces obtained by steam operation,

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the maximum being determined by the steam pressure which can be obtained in the cylinder. With motors of hoisting types, torques can be obtained which are so far in excess of anything that the steam engine could give that this sometimes results in disaster when a motor is applied to a hoist of that type.

If the operator gets a little careless, he may overstrain the hoist, and cause damage, so that this question should be looked into when changing from a steam hoist to an electric hoist. The hoist should be rugged enough to stand the torques which may be attained by careless operation, in which case, instead of using a three-bearing motor, with a pinion between two of the bearings, it is much better to arrange for a two-bearing motor, arranging for a coupling through a flexible coupler.

In another place trouble is mentioned as due to the vibration of a geared hoisting motor. That can be overcome by supplying a flexible coupling between the motor and the hoist proper. You will find, if a flexible coupling is furnished, that the motor troubles will largely disappear.

K. A. PAULY, Schenectady, N. Y. (communication to the Secretary*). —This paper is deserving of careful consideration by all interested in general electrical mining problems and the recommendations made should be given weight because of the author's long experience in dealing with various problems encountered from the beginning of mine electrification.

In discussing the paper I will simply make note of some of the more important features in which American practice differs from that of South Africa. To do more than this would be amiss, as each of the subdivisions of Mr. Bulkley's paper might readily be made the subject of a lengthy paper in itself.

We have read much recently of high-pressure centrifugal compressors and I would like to hear a little further from the author with reference to the general results obtained with the large units installed at the Robinson Central air station; how they compare in efficiency and cost of operation (fixed charges and operating costs) with the reciprocating types, and what methods, if any, are used to control the output with varying demands for air.

It is interesting to note the difference between South African and American practice in tube-mill drives. It is general practice here to gear motors rather than belt them to the mills, and to use squirrel-cage type motors rather than the slip-ring type for this service, and as far as we have been able to learn, good results have thus far been obtained; in fact, much better than I expected, as I rather favored the slip-ring motor. We have not had any considerable difficulty in obtaining suffi-

* Received Feb. 11, 1916.

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cient starting torque, although the motor must be especially designed with this in view. A factor which influences me to favor slip-ring motors is the possibility of injuring the stator windings of high-torque squirrel-cage motors in starting by the vibration of the portions of the coils external to the slot due to the heavy currents during starting. I would like to learn whether or not squirrel-cage motors were given a thorough test in South Africa before the adoption of the slip-ring type and, if so, for how long a period and what troubles were experienced.

It is interesting also to note the difference from our practice in motor-driven pumps in that the motors are belted to the pumps rather than geared, as is common practice in this country. We have experienced some trouble from motors geared to reciprocating pumps, but have found that in practically all cases the trouble was traceable to faulty installation, usually a lack of rigidity in the mounting of the motor, which, unless special precautions are taken, may readily be the case with vertical pumps. Of course, the motor must be of thoroughly rugged construction, but where the installation has been properly made little or no trouble has been reported.

In connection with high-voltage motors underground I wish to point out that recently we have made some radical improvements in our insulating materials so that we feel entirely justified in installing motors of 2,300 volts underground, unless the conditions surrounding the operation of the motors are especially severe, in which case, either low-voltage motors must be used or the conditions improved. In this connection it must be borne in mind that it is not practical to make the factor of safety in high-voltage installations as great as is readily obtainable with low-voltage installations, but with the new special-treated moisture-resisting insulations 2,300-volt motors installed underground give reliable service and these motors are standard today with many companies for underground service.

About the middle of p. 144 we note that liquid-rheostat control is used with alternating-current mine-hoist motors as small as 100 hp. The practice in this country is to use magnetic control for motors as large as 400 hp., this control being lower in first cost for these sizes and thoroughly reliable.

At the bottom of p. 145, in connection with unbalanced hoisting, our experience has rather indicated that with alternating-current hoists, especially for long lifts, the torque required for unbalanced hoisting is the limiting feature rather than the heating, although for shorter lifts where the ropes play a less important part, heating may readily be the limiting factor. With direct-current hoists with flywheel motor-generator sets, to meet the unbalanced condition at full load and full speed a prohibitively large flywheel is frequently required. Usually a reduction in the load is permissible when hoisting unbalanced.

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I am much interested in the author's comments regarding the eddy-current brakes as referred to under "Mine Hoists," as I have always felt as Mr. Bulkley does with reference to these devices.

In connection with the author's comments, at the foot of p. 148, on the maintenance cost of electric winders I take the liberty of quoting from a letter received from one of the pioneers of electric hoisting in this country.

"As to the relative merits of hoisting with electricity as compared with steam, I would say unqualifiedly that the electric hoists are more reliable than steam hoists, granting that you have, of course, a constant and reliable current. The electric hoists are more sensitive to handling, are much easier controlled and it is easier to apply overwinding and other safety devices to them. The electric hoist gives a steady pull on the rope during the acceleration and, in fact, during the entire trip. There is none of the flopping of the rope which is apparent in the pull of a steam hoist. We find that for this reason our pulley stands, which support the rope between the engine house and the shaft house, can be set much farther apart, and that the rope never jumps out of the groove in the carrying sheave."

J. NORMAN BULKLEY, New York, N. Y. (communication to the Secretary*).—Referring to Mr. Pauly's question regarding general results obtained with the large turbo compressors, I have never seen any figures on maintenance costs for them. Judging, however, from the operation of the station, the results should be good. There are no governing devices, the output being controlled by blowing-off and shutting-down units.

Tube-Mill Drives.—I have had no experience with the high-torque squirrel-cage motor and so far as I know, the ordinary squirrel-cage motor has never been used in South Africa, owing entirely to trouble in starting. I believe that heavier pebble loads are carried in African practice, which may account for the difference.

Hoisting.—In speaking of heating as the limiting factor for unbalanced hoisting I had in mind only inclined shafts. Of course, I agree with Mr. Pauly as to the necessity for reducing the load and possibly the speed when hoisting unbalanced with flywheel sets.

* Received Mar. 10, 1916.

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Researches on Fire-Damp

BY ENRIQUE HAUSER,* MADRID, SPAIN

Translated from the French by Bradley Stoughton and G. A. Burrell

(New York Meeting, February, 1916)

FIRE-DAMP is a mixture of methane with other inert gases or combustible gases. The inert gases in question are carbonic acid, water vapor, nitrogen, etc. The combustible gases are hydrogen, ethane, etc. The study of the properties of fire-damp, therefore, becomes a study of the properties of methane and the influence on its properties of the different gases we have mentioned.

Characteristic Properties of Methane

1. *Respirability.*—Methane being a paraffin is characterized by its weak chemical affinities. From the physiological point of view one could substitute it for nitrogen of the air at least momentarily, without experiencing any difficulty in breathing the mixture consisting of 21 per cent. O_2 + 79 per cent. CH_4. I have tried this on myself, making the gas exhaled from the lungs burn subsequently in a Bunsen burner.[1] From the chemical point of view, this weak affinity permits the gas to be burned in certain circumstances with hydrogen and oxide of carbon in an oxygenated mixture, with the result that the last two gases achieve only a fractional combustion. This practice has been utilized with success in the methods of analysis which I have perfected and which I describe later in this paper.

2. *Retardation of Flashing Point.*—A characteristic property of methane is its retardation of flashing point, noted by Davy in 1816 and studied by Mallard and Le Chatelier in 1883.[2] Taffanel and Le Floch have recently confirmed these results.[3]

Let us recall the nature of this retardation:

When a mixture of combustible gases with air or with oxygen is

*Professor, School of Mines, Madrid, Spain.

[1] El Grisú en las Minas de Carbón.—*Primera Conferencia Experimental*, 1907. Leçons sur le grisou, *Première Conférence Expérimentale*, 1908, pp. 57–58.

[2] *Annales des Mines*, 1883, pp. 274–296.

[3] C. R. de l'Ac de S., 19 Mai, 1913.

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submitted to the action of heat, combustion commences and continues slowly without any external manifestation indicating the occurrence of a chemical reaction. But, if we raise the temperature little by little a point is reached where this combustion takes place rapidly, manifested externally in the phenomena of light and finally in the diminution of volume of the mixture. It is the occurrence of these latter phenomena which is called the flashing point of the gas. Thus, according to the experiments of Mallard and Le Chatelier, in a mixture of carbonic oxide and oxygen, combustion commences with appreciable rapidity at a temperature of 400° C.; at 447° C. the proportion of the mixture which

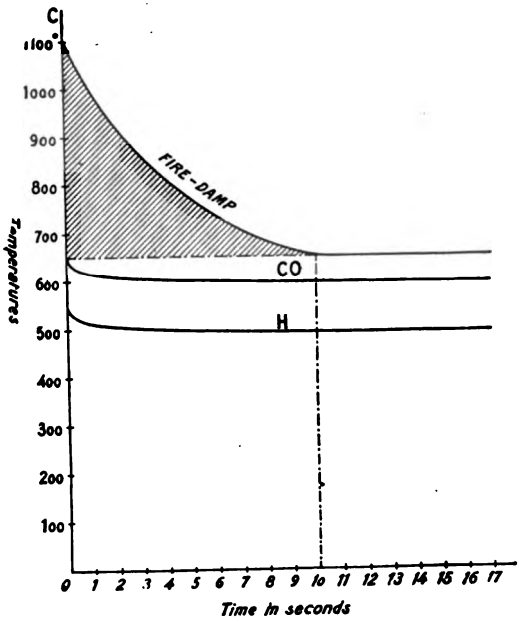

FIG. 1.

burns in a second is 1 in 1,000; at 615° C. the proportion is 1.5 in 1,000; and, at 650° C., the flashing point is immediately reached. That is to say, this great change occurs in a temperature interval of only 35°. At 650° C. the whole mixture is burned in some thousandths of a second.

The same general result is noted in the case of hydrogen: For example, in an experiment made at 540° C. only one-half of the mass was burned, while at 555° C. combustion was completed on the instant.

In mixtures of methane with air these phenomena of retardation are very much amplified: Slow combustion commences at 450° C. without arriving at the flashing point. More rapid combustion occurs at 650° C., requiring about 10 sec. to accomplish the burning of the mixture. This

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duration, or this retardation, becomes almost negligible at 1,000°C. In mixtures of fire-damp and oxygen, the retardation of the flashing point is a little less.

In order to represent these phenomena better I have drawn a curve (Fig. 1) where the ordinates represent temperatures and the abscissæ represent the duration in seconds. The value of the abscissæ will then indicate to us the time required for the gas in question to reach the flashing point at any given temperature. In the figure herewith are represented the curves of hydrogen, carbonic oxide, and methane, which cut the axis of ordinates at the temperature of the flashing point without retardation, and which are asymptotic to the axis of abscissæ at a distance corresponding to the temperature at which slow combustion commences. The part shown in cross-section at the interior of the curve for fire-damp corresponds to the combustion which is terminated at the flashing point, or, in other words, the breaking into flame.[4]

The scientific reason for the retardation of the flashing point should be sought, in my opinion, at least in part, in the fact that methane is an exothermic gas which requires for its dissociation 22.1 calories per molecular weight, and which does not arrive at the flashing point until the moment of dissociation. The necessary calories must, therefore, be supplied by the combustion of a part of the same gas previous to its breaking into flame. In order to give an idea of this we will say that the dissociation of 100 liters of methane requires a quantity of heat equivalent to that produced by the combustion of 32 liters of hydrogen. If this is exact, then a mixture of methane and hydrogen containing 25 per cent. of hydrogen should reach the flashing point without any retardation. In trials which I have made where heat was applied at a single point, the quantity of hydrogen actually necessary was 7.2 per cent. higher.

Be that as it may, it is necessary to consider this phenomenon of retardation of the flashing point as a protection which hides a danger and which has often been the cause of serious accidents, because of too great confidence reposed in the resistance to the flashing point of fire-damp. The object of my first researches on this gas was to reach the flashing point of methane by means of incandescent wires. During this study the phenomenon of retardation became manifest.

3. *Flashing of Fire-Damp by Incandescent Wires.*—The flashing of fire-damp by means of metallic wires heated to a red heat by an electric current has been the source of many discussions, but, if one studies this question profoundly, it becomes evident that what appeared at first to be divergencies in reality do not exist as such. In order to study the flashing by means of metallic filaments it is doubtless necessary to employ

[4] El Grisú en las Minas de Carbón, *Primera Conferencia Experimental*, 1907, p. 19.

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a current of low tension and a circuit without induction, in order that, at the moment of fusion of the metallic wire, sparks shall not be produced by the breaking of the circuit, because these sparks would, of course, produce the flashing of the mixture.

That is why I employed in my experiments a current of 4 volts, obtained from two accumulators with lead bases of the Dinin model, placed close to the metallic wire in order to avoid induction effects from long conductors. Under these conditions it is not possible to produce appreciable sparks upon rupture. They would have occurred if I had employed a resistance in series to absorb the voltage during the dynamic state of the passage of the current, as was often the case in the experiments of my predecessors on this subject. On the contrary, in the experiments described below I have never obtained an explosion by rupture of the wire and subsequent sparking.

But there is a very important cause which has contributed without doubt to the apparently contradictory results of Couriot and Meunier, because these experimenters, according to their own statement, worked constantly on the mixture of 9.5 per cent. of fire-damp.[5] Now, as the higher limit of the flashing point of fire-damp is 14.8 per cent. of fire-damp, according to the experiments of Burgess and Wheeler, the mixture used by Couriot and Meunier contained an excess of air of only 14.8 − 9.5 = 5.3 per cent., which is only 1.05 per cent. of oxygen more than the content required for the higher limit of flashing point. If this 1 per cent. of oxygen should be used up by oxidization of the wire there would be left a non-inflammable mixture. This is the reason why Wullner and Lehmann found the most diluted fire-damp mixtures (6.66 per cent.) the most easily inflammable. It is on these diluted mixtures that I made my experiments by employing at times methane prepared in the pure state by means of carbide of aluminum, or better, natural fire-damp.

Under these conditions I have obtained the following results:[6]

1. With wires of ferro-nickel, 0.3 mm. in diameter, with or without fusion of the wire, I did not obtain inflammation of the most sensitive mixtures of pure methane.

2. With platinum wires, 0.5 mm. in diameter, gradually brought to a red heat, I have obtained inflammation in six consecutive trials with mixtures of from 7 to 7.5 per cent. of natural fire-damp, without seeing the filament melt, although it glowed rapidly at the moment when the explosion took place. With filaments of platinum of 0.2 mm. diameter, I have produced two explosions in three experiments with natural fire-damp.

[5] H. Couriot and J. Meunier: Recherches sur l'inflammabilité électrique des mélanges d'air et de grisou, p. 12, 1906.

[6] El Grisú en las Minas de Carbón. *Primera Conferencia Experimental*, 1907, p. 26; Leçons sur le grisou, *Première Conférence Expérimentale*, p. 28.

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3. With wires of soft iron, of 0.9 mm. diameter, the results obtained are very interesting: In employing a straight filament, either horizontal or inclined, or a filament curved toward the top or toward the base, I have obtained five inflammations in 17 experiments with natural fire-damp of from 7.2 to 7.5 per cent. That is to say, one-third of the experiments resulted in inflammation without fusion of the filament. (In cases where inflammation did not result the filament was, however, heated to fusion.) On the contrary, by employing an iron wire inclined at an angle of nearly 45°, with a spiral tower toward the middle, I obtained five inflammations in five experiments, without fusion of the filament, and in three of these experiments I employed the same filament for three consecutive attempts. In another case I inflamed with a filament of twisted wire a mixture which had not been inflamed by fusion of the straight wire. The explanation which I offer for these facts is as follows:

It will happen in the case of a relatively large filament of iron that its center will be at a higher temperature than the outside, which becomes rapidly covered with a layer of melted oxide of iron. Now, if a point exists where the cohesion is somewhat greater than elsewhere, and where the oxide runs together in the form of a ball or a pearl, the center of the wire will be exposed, and the filament will be partially volatilized and become an oxide in the state of vapor, and will produce a flame which will set off fire-damp. Such an event could happen with a galvanized iron wire at the temperature of the volatilization of zinc, which is 675°—or almost 1,000° lower than the temperature of fusion of soft iron. On the other hand, if we expose a slight obstacle to the movement of the fire-damp, for example, by placing a cold filament on the incandescent filament, we would be able to produce the explosion more easily, and we can produce it with certainty if by rolling the wire in the form of a helical we heat the gas from two sides and require it to pass at least twice across the red-hot filament. In this case the explosion occurs at a temperature lower than in the case of a straight filament.[7]

[7] Couriot and Meunier (*Annales des Mines de Belgique*, 1908, vol. 1, p. 90 to 91) thought that this explanation should be rejected, because they could not admit that, at a temperature lower than that of its fusion, "iron would be volatilized and, oxidizing in a state of vapor, produce a flame," and that, even admitting as possible that the volatilization and the flame took place, it would be as well produced with a straight wire as with a curved wire, and better still with a small than with a large wire. With all respect due to these gentlemen, I submit on this point that, being obsessed with their theory of a gaseous envelope of oxygen, they did not well understand my experiments, which often resulted in a contradiction of that theory. I do not think it necessary at the moment to discuss extensively the theory of Couriot and Meunier, but I would say that it is not possible to admit that fire-damp can be inflamed by radiation alone, because the inflammation must commence at the hottest point, that is, at the contact with the incandescent wire. Now, as a single contact does not suffice, because of its short duration, one should curve the wire in the form of a helical and incline it, to prolong this contact and transform the slow combustion of the fire-damp into a rapid

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4. To confirm these experiments I made others with soft-steel wires 0.6 mm. in diameter and with pure methane from pure carbide of aluminum. In four attempts I did not obtain inflammation with a straight, horizontal filament of about 15 mm. length, but inflamed once with a filament about 25 mm. in length and curved toward the top. With an inclined filament, having three helical turns, I obtained two explosions in two experiments, using the same mixture with which I failed to obtain an

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combustion, that is to say, a combustion accompanied by flame, which combustion is, however, only visible at a certain distance from the wire, where there is an unburned part of the mixture. Furthermore, as I have observed previously, the temperature of the curved wire at the moment of the inflammation of the fire-damp is lower than in the case of the straight wire. It is between clear orange and white, instead of welding white or dazzling white. This, therefore, makes the radiation less important and makes entirely improbable the production of a flame by volatilization of iron of the center, which is not exposed as in the case of a straight wire heated to a dazzling white heat. I should remark here that neither the length of the wire employed by me nor the duration of my experiments permitted in any case the absorption by the wire of oxygen from the explosive mixture, so as to render it non-inflammable. The amount of fire-damp in these mixtures was never higher than 7.5 per cent. as I have already stated.

As to the theory of the possible volatilization of the center of large iron wires, which has been suggested by the observation of the easy inflammability of fire-damp by a galvanized iron wire brought only to a dark red heat, I cite the following facts: First, it is without doubt that a large wire, heated by an electric current, will have a higher temperature in the interior than on the surface, because of the loss of heat by radiation and convection. It is because of this fact that Heraeus has substituted in his electric furnace platinum ribbon instead of platinum wire, because the wire becomes brittle on account of the high temperature to which its center is submitted, and because this part is most distant from the chamber of the furnace. On the other hand, it is also a fact that because of the great difference in the temperature which exists between the center and the surface of steel ingots in metallurgical works that the molds are stripped in order to roll the steel, without first passing through a heating furnace. To this end the ingots are introduced into soaking pits, which are not heated from an exterior source. It is only necessary to establish an equilibrium of temperature in the mass of the ingots before they can be taken directly to the rolling mill.

In reality, in the case under discussion, the difference of temperature between the center and the surface of a wire is not very great, because oxide of iron melts at 1,350° C. (Le Chatelier-Boudouard, *Mesure des Températures Elevées*, 1900, p. 167), and its gradual formation should liberate enough heat to compensate in part for the superficial loss by radiation and so cause the hottest zone to be found near the surface. Now, this layer of oxide, gradually formed, prevents in its turn the rapid oxidization of the wire. If now this layer should suddenly disappear by the accumulation of oxide at one point, the center of the wire would be exposed and, since it is already near its point of fusion, its superficial oxidization would take place in this case very rapidly and the heating would be sufficient to melt it, with partial volatilization. This phenomenon of partial volatilization of iron by its heat of oxidization is comparable to that which takes place toward the end of the second period of the Bessemer process.

I think that these explanations, once given, enable one to admit the possibility of the hypothesis which I have formulated to explain the inflammation of fire-damp by a large, straight, incandescent wire.

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inflammation in the three consecutive attempts with the straight, horizontal filament of 15 mm. length. This last experiment was repeated with the same result, employing natural fire-damp.

I think that the results could not be more conclusive, and that, if the flame or the electric spark were the most appropriate means for inflaming fire-damp, it is nevertheless true that incandescent filaments could produce inflammation with equal certitude without the intervention of the flame, on condition that the wires are not melted during the time of the retardation of inflammation of the fire-damp, and provided that the

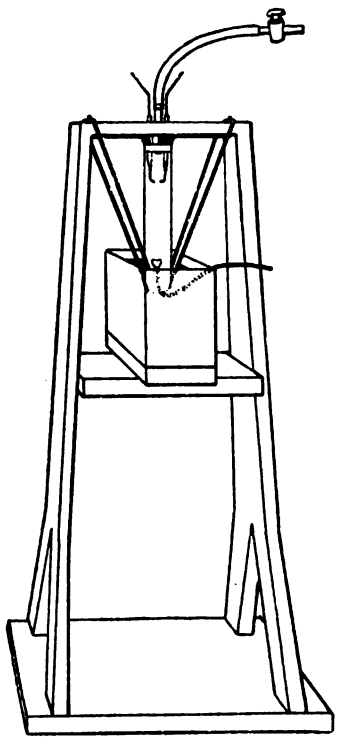

FIG. 2.

temperature of the wire and its heating surface are such that the fire-damp mixture in contact with it can be rapidly brought to the temperature of quick combustion or of inflammation, before it burns slowly or its oxygen is absorbed by the incandescent wire.

The apparatus which I employed for the experiments of inflammation consists, as shown by Fig. 2, of a large cylindrical glass tube of 40 mm. diameter and 155 mm. of useful length,[8] placed in a bowl filled with water. It is closed at the top by a rubber stopper perforated with three holes.

[8] Total length, 180 mm.

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A glass tube passes through the center hole and is joined to a pet cock for introducing the gaseous mixtures. Glass tubes also pass through the other two holes and terminate in metallic points at their lower ends, between which the spark can be produced. By means of a drop of mercury the connection is established between the metallic points and the wires of the exterior circuit which enter the tubes. By adjusting the height of the tubes and points in the cylinder the spark can be regulated at the desired height, but, to make the experiment more sure, I employed sometimes two pairs of points, one at the upper part of the test tube and the other toward the base. To try inflammation with incandescent filaments, a curved cable with two conductors, duly isolated so that their points are separated in the form of a fork, is introduced at the lower part

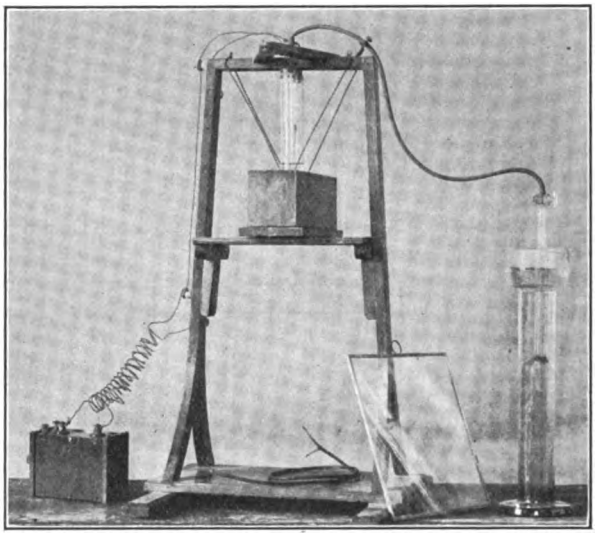

FIG. 3.

of the test tube through the bowl of water. Between the points of these conductors is placed the filament which is to be brought to a red heat by the passage of the electric current. The filament is situated about one-third or one-half of the distance from the bottom of the test tube. The entire system is bound together by iron stays on a solid support of wood, as shown in Figs. 2 and 3.

The results of these experiments have also proved to me that pure methane prepared in the laboratory conducts itself in the same manner as natural fire-damp as to its inflammation by incandescent filaments, from which I have been led to believe that it can be used with an incandescent wire of small diameter and serve as a physico-chemical means of characterizing the influence which other combustible gases have on natural fire-damp.

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4. *Influence of Combustible Gases on Properties of Fire-Damp Mixtures.* —Many discussions have been had on the existence of hydrogen and hydrocarbons in fire-damp, as indicated by chemical analysis. The discussions have involved especially the consequences which the presence of these gases, more inflammable than methane, would have upon our knowledge of the properties of so-called "sharp gas." For my part, I believe that, instead of entering into academic discussions based on samples of gas which it is impossible to procure twice under the same conditions, it would be much better to study the properties of synthetic fire-damp consisting only of pure methane, and then to examine the influence exercised upon the properties of this gas by the addition of variable quantities of the combustibles mentioned above, in order to be able to determine at just what point their mixture with fire-damp would augment the danger.

This, however, has not prevented me from studying the question of the analyses of fire-damp, of which I will speak later.

The inflammable gases which could accompany methane in fire-damp, through circumstances which are fatuitous or a little understood, are ethane, ethylene, and hydrogen.[9] None of these gases have an appreciable retardation of the flashing point, so that it would be easy for us to study their influence on fire-damp by preparing mixtures of each of them with methane in different proportions and seeking to inflame them with metallic wires of a diameter which would be incapable of inflaming methane alone.

From the conclusion that the most oxygenated mixtures are the most easily inflamed, I have always attempted in my experiments to maintain mixtures close to the lower limit of inflammability, without however always accomplishing it. I have obtained the following results, which I consider as simply a step in advance, but which give an idea of the influence exerted by the presence of other gases on fire-damp mixtures.

Of ethane, whose limit of inflammability was 3.9, I employed 4.5 parts with 1.82 parts of methane, in order to accomplish inflammation by the fusion of a ferro-nickel wire of 0.3 mm. diameter. In this case the volume of ethane was 66 per cent. of the total volume of the two inflammable gases.

Of ethylene, whose limit of inflammability was 3.6, I used 4 parts with 2.85 parts of methane to obtain inflammation under the same circumstances. In this case the ethylene was 58.5 per cent. of the total volume of the two inflammable gases.

Of illuminating gas, with a limit of inflammability of 8.5, I obtained inflammation using 5 parts mixed with 4.5 parts of fire-damp, the illu-

[9] El Grisú en las Minas de Carbón. *Primera Conferencia Experimental*, 1907, p. 44.

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minating gas being, therefore, 54 per cent. of the volume of the two inflammable gases.[10]

With hydrogen and methane I obtained inflammation with a mixture containing 2.9 parts hydrogen and 6.1 parts fire-damp, the hydrogen therefore representing 32.2 per cent. of the total volume of the inflammable gases.

Thus we see that a large quantity of the foreign gas is necessary to cause methane to lose its retardation of flashing point under the conditions existing in my experiments. This does not prevent a relatively small quantity of these same gases from extending the limits of inflammability of fire-damp mixtures, but it is interesting to note that methane conducts itself in this case like a true paraffin in diminishing the sensitiveness of the inflammable mixtures, as is the case with gun cotton impregnated with paraffin, which requires a much stronger detonator to detonate it than if it were in the dry state.

The relatively large quantity of these gases, which it is necessary to add to pure methane to render it inflammable by means of a thin metallic incandescent wire, proves that the presence of traces of hydrogen or of ethane reported by chemical analysis would not have any appreciable influence on the characteristic properties of fire-damp. In any event, this natural gas should not be protected by a "sacred taboo," which would take away the value from experiments made with methane sufficiently pure, and which can be prepared at a low net cost by a method which I will indicate later. On the contrary, the possible influence on the properties of natural fire-damp in fuel gases of hydrogen and oxide of carbon should be taken into account.

5. *Influence of Inert Gases on the Properties of Fire-Damp Mixtures.*— The influence of nitrogen and carbonic acid on the limits of inflammability of methane is only appreciable as the diminution in the proportion of oxygen is felt. My experiments, in accord with those of Le Chatelier, show that each 1 per cent. of carbonic acid added to the air up to 8 per cent. raises the lower limit of inflammability by about $\frac{1}{1000}$. But the case is a little more complex in mines where the gas coming from the operation of gobbing produces an enrichment of the air in the mines in carbon dioxide and water vapor at the expense of the oxygen of the air. The upper limit of inflammability descends rapidly and the lower limit is raised slowly in direct ratio as the content in oxygen is diminished, until we have only a single value toward 13.5 per cent. of oxygen with a content of 6.7 per cent. of methane. Without going further in this discussion I should say that I have abandoned the studies made by me in this way after having

[10] It is appropriate to compare here the resemblance of these results with illuminating gas to those reported by the French Commission on Fire-damp in its experiments on the inflammability of fire-damp by sparks obtained from a blow of steel on stone.

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noted the exhaustive work done in the United States by J. K. Clement.[11]

But there is still another interesting point to note; namely, the influence of carbon dioxide on the diminution of the rapidity of combustion of fire-damp mixtures, of which it forms a part and from which one can draw material for useful deductions. I will recall first that, as noted by F. Clowes,[12] the composition of air when it leaves the flame of a candle or of an oil or alcohol lamp is comparable to that exhaled from the lungs. Now, since the gas from gob areas is also the result of a slow combustion, it is easy to see that the air exhaled from the lungs could be substituted in the laboratory[13] in experiments with fire-damp, of which those which follow have a special interest:

Let us now consider the relighting, without danger, of safety lamps in a fire-damp atmosphere.[14] The relighting of safety lamps in a mine has been the cause of many discussions and Marsaut[15] has said relative to materials for relighting by friction, which are considered the safest today (as based on the experiments at Frameries on this subject):

"But for those (relighters of white phosphorus) the experiments at Frameries and elsewhere do not give the certainty that the lamps submitted to the proof were completely freed from inert gas before the relighting. At Frameries prompt relighting with wire gauze made red hot or very warm especially occupied attention, but under these conditions could one be sure that the lamp is free from all inert gases of former combustion at the moment of relighting? The very capricious results of the tests allow one to doubt, or even to think the contrary.

"The non-explosive phosphorus relighters which are supposed to be almost inoffensive, were not submitted at Frameries to a sufficient number of tests to lead to a sure conclusion."

In consequence of the above I am led to believe that, if we could assure ourselves that the atmosphere of the lamp at the moment of relighting was under the same conditions as when we studied this relighting, the problem would be solved. Now, the gas exhaled from the lungs mixed with fire-damp diminishes considerably the latter's rapidity of combustion, which becomes comparable to that of mixtures near the limit of inflammability. Then, if in a safety lamp which has been extinguished some time and which no longer contains inert gases of combustion we inject air from the lungs, one would then be able to fill the lamp with a fire-

[11] *Technical Paper No. 43 of the U. S. Bureau of Mines.*

[12] The Detection and Measurement of Inflammable Gas and Vapors in the Air, p. 168.

[13] Air exhaled from the lungs has the following composition:

CO_2 = 4.4 per cent. N = 80.4 per cent. O = 15.2 per cent.	Equivalent to	CO_2 = 4.4 per cent. N = 22.9 per cent. 57.5 per cent. O = 15.2 per cent.	= 27.3 per cent. (black damp) = 72.7 per cent. (air)

(*Primera Conferencia experimental*, 1907).

[14] El Grisú en las Minas de Carbón, 2ª *Conferencia Experimental*, 1908, p. 125.

[15] *Bulletin de la Société de l'Industrie Minérale*, Mar. 21, 1907.

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damp mixture which is inflammable, but only feebly explosive, and non-capable of propagating the flame to the exterior.

After that it would be easy to judge the effect of the exhaled air on the combustion of a safety lamp by blowing softly on the top, an operation which could be executed with very little practice. From another point of view I have been able to observe with an acetylene safety lamp different phases of the extinction of fire-damp mixture through its impoverishment in oxygen and its mixture with residual gases of its own combustion, by working as follows: To make this experiment, which has never been published before, I used a lamp of the Tombelaine system with low admission, which permits the flame to be reduced at will, to examine the aureoles.[16] If now, having reduced the flame to merely a luminous point, we introduce the lamp under a glass bell of about 15 to 20 liters capacity, half filled with methane or fire-damp, and gradually raise the lamp, we see the cap first elongate and the inflammation *en masse* of the gas contained within the gauze. But instead of seeing the flame extinguished, as would be the case of an acetylene flame thrust into an atmosphere which contained 12 per cent. of oxygen,[17] we see the gas first inflame and then become merely extinguished at the wick; then as the air entering at the bottom of the lamp dilutes the products of combustion, we see the acetylene flame form a new aureole which is elongated in its turn with inflammation and new extinction of the fire-damp mixture in the same manner as before; and thus we could repeat the same thing several times, if desired.

These experiments allow an answer to be made to a question often asked by mine foremen, namely, why one sometimes sees fire-damp suddenly inflame at the interior of lamps, accompanied by an explosion, while at other times there is formed only a very much elongated aureole without sudden extinction. The answer is easy: If the air of the mine is pure its mixture with fire-damp is at first explosive, but, if the air of the mine is rich in black damp, as the residual from respiration or from its mixture with gas from gobbing, the aureoles are more or less elongated and the inflammations in the interior of the lamp do not become true explosions.

I think that an extension of these considerations would give us an explanation of "sharp gas."

Preparation of Pure Methane by Means of Commercial Carbide of Aluminum

Admitting that the properties of fire-damp as an inflammable gas could be studied starting with pure methane, it remains to explain the

[16] Tombelaine: *Informe sobre la lámpara minéra de seguridad de Acetileno sistema*, 1912.

[17] 11.9 of O_2 + $7CO_2$+81.1 N_2 (according to Beyling). It is interesting to note in this connection that methane will not burn when the mixture contains less than 14 per cent. of combustibles.

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method of preparing this gas under conditions of purity necessary to this end and at a price sufficiently reasonable for the purpose.

My predecessors in these studies have not made sufficiently clear the fact that methane prepared by means of the acetate often contains unsaturated hydrogen and hydrocarbons,[18] and that prepared by means of the commercial carbide of aluminum may contain considerable quantities of hydrogen, especially if it is produced hot.[19] It remains to find other means to purify the gas, which would require a second operation, or to employ pure carbide of aluminum prepared by the Moissan process, costing 400 francs per kilogram. This would make a cubic meter of pure methane at 0°C. and 760 mm. pressure cost about 1.430 francs. For these reasons I have undertaken another study in which I have succeeded in preparing pure methane by means of commercial carbide of aluminum which only costs 3.30 francs per kilogram.[20]

The principal impurities of carbide of aluminum are the carbides of calcium, iron and silicon, together with an excess of aluminum and uncombined carbon and of alkalies with a little sulphide of aluminum, and perhaps silicide of aluminum.

The carbides of iron and of silicon are not attacked by cold water so that we are not handicapped by their presence. As carbide of aluminum is only attacked slowly by cold water the problem resolves itself into the separation of the carbides of calcium and the alkalies by a simple treatment with cold water, provided the hydrates of lime and alkali liberated by this reaction, and which remain inclosed in granules of carbide of aluminum, do not later act on the metallic aluminum in excess, which the carbide contains, and produce hydrogen, because it is this gas mixed with methane which modifies most strongly the properties of the latter gas. But even in the absence of alkalies, aluminum decomposes water slowly by virtue of the following thermic equation:

$$Al_2 + 6H_2O = Al_2O_3.3H_2O + 3H_2 + 186 \text{ calories.}$$

This reaction is more easily disturbed if the aluminum contains traces of calcium or alkali metals. On the other hand, the reaction once commenced will be immediately lessened in rapidity on one side by a slight layer of hydrate of aluminum which covers the metal, and especially by the presence of little bubbles of hydrogen which cover it. These are easily disengaged either by heating or by forming a vacuum above the liquid. By taking note of the different facts which I have exposed, I have deduced the following method for the purification of carbides:

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[18] See Leçons sur la Grisou, *Première Conférence Expérimentale*, pp. 13 and 14.

[19] El Grisú en las Minas de Carbón, *Segunda Conferencia Experimental*, p. 9, *et. seq.*

[20] For more details of this process of purification see *Segunda Conferencia Experimental*, pp. 7 to 21, and a note published in *Anales de la Sociedad Española de Física y Química*, May 5, 1913.

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It is necessary first to take out the excess of aluminum and this can easily be done by pulverizing the carbide between the cylinders of a rolling mill which, at the same time, flattens out the metallic aluminum. It then only remains to sift the mixture to separate the good part from the metal.

To get rid of the carbide of calcium, the sifted carbide of aluminum is treated several times with water until the supernatant water is clear. As the hydrate of calcium is only slightly soluble in water we transform it into a very soluble chloride by means of washing with dilute hydrochloric acid. For this purpose use 5 per cent. of acid of 1.19 specific gravity, then continue washing with water until the wash water no longer is acid. For 50 grams of carbide we would require about three washes with 500 c.c. of water before the treatment with acid and as much again afterward.

After this purification the carbide is in condition to be used, but one should also add the precaution of rapidly drying by carrying off the water by means of alcohol followed by one washing with ether if the carbide is to be kept.

In order to obtain from this carbide a methane almost free from hydrogen it is necessary to avoid as much as possible the disengaging of little bubbles of hydrogen which cover the aluminum during the attack on the carbide. This can be done by giving a certain thickness to the mass of carbide on the bottom of a conical flask where the reaction is produced. If, on the other hand, we give a certain height to the water which the gas has traversed to reach the gasometer one will then find it in its best condition and one can obtain in winter, with the temperature about 14°C., a methane almost free from hydrogen (about 0.2 per cent.).

But this process becomes incomplete in summer where sometimes I have found in the methane as much as 8 per cent. of free hydrogen. It is easy to see that under these conditions the liquid has become alkaline, its analysis showing it to contain aluminate of soda.[21]

To prevent this occurrence it is sufficient to replace daily the water which covers the carbide by washing until the wash waters are no longer alkaline. Operating thus we can obtain even in summer methane containing less than six parts per thousand of hydrogen, which is available for the greater part of the experiments. For the work of analysis it suffices to purify the gas by washing it with potash and passing it through a tube containing palladium.

[21] For further details on this subject see my "Nota sobre la obtención del metano puro mediante el carburo de aluminio comercial." *Anales de la Sociedad Española de Física y Química*, May 5, 1913.

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Discussion

George A. Burrell, Pittsburgh, Pa.—Mr. Hauser determined the influence of inert gases on the properties of fire-damp mixtures, and one might take some issue with his work on the effect of hydrogen, on those mixtures. It is valuable information to have, but as far as the practical application of it goes, it is not of much significance, because one does not find it in mines, at least not in any appreciable amount. The same

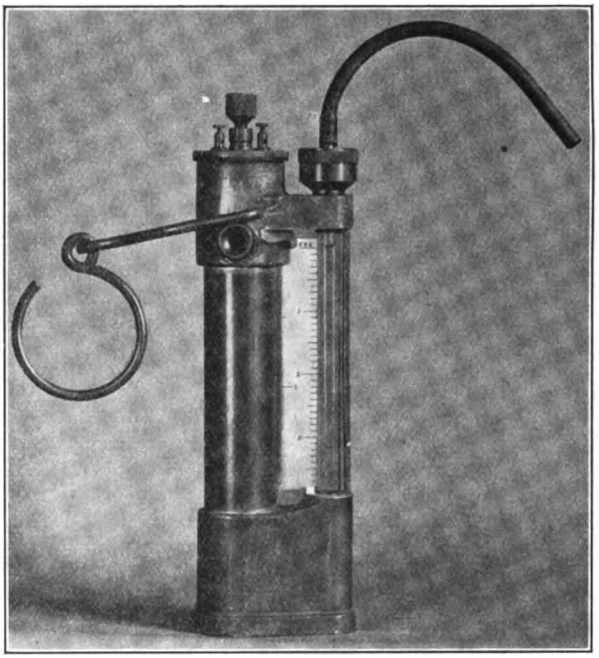

Fig. 1.—Apparatus for Detecting Combustible Gases in Air.

thing can be said of ethylene. Ethylene is formed in considerable quantities as a product of heat reaction, but normally, in the coal mines, as a result of a great many analyses we have made at the Bureau of Mines, we cannot say that ethylene is a combustible gas found in the fire-damp in coal mines.

The author winds up his paper by a very good method for preparing methane. I have not tried that method myself, but it has been used by the British experimenters at Altofs. We have a rather simple means of getting methane ourselves. About 10 miles out of the city there exists a natural-gas well under about 200 lb. per square inch pressure, containing 98.8 per cent. of methane, a trace of carbon dioxide, and about 1 per cent. of nitrogen.

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There is a device, which we have recently developed at the Bureau of Mines, for detecting combustible gases in air.

Fig. 1 shows a photograph of the apparatus. It is made of brass and aluminum, except the stout glass gage at the right, and weighs 1.4 lb.

A diagram of the instrument is shown in Fig. 2. It consists essentially of a U-tube of which the parts *A* and *B* are two limbs. To make the machine ready for use one unscrews the head piece *C* and pours water

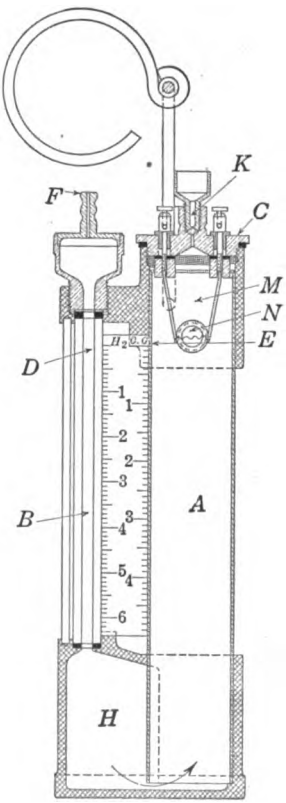

FIG. 2.

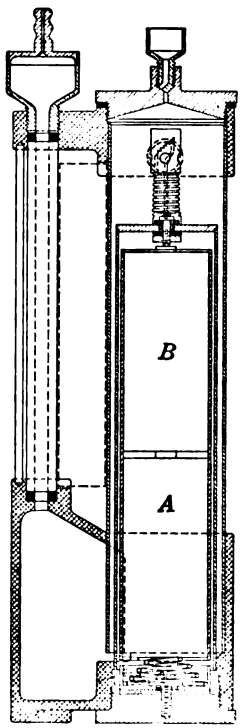

FIG. 3.—ANOTHER FORM OF APPARATUS FOR DETECTING COMBUSTIBLE GASES IN AIR.

into the tube *A* until the water rests at the point *D* opposite the zero mark of the scale. The water will then rest at the point *E* in the brass tube *A*.

To collect a sample of mine air for test one blows into the tube *F* by means of the mouth, thereby depressing the water in the glass tube *B* to some point in *H* and raising the water in the other tube *A* to the valve *K*. Next the water is allowed to fall to its original position (levels at *D* and *E*) thereby drawing a sample of the mine air into the combustion

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chamber *M*. Then the small platinum spiral is electrically heated, thus burning methane that may be in the sample. Methane burns as follows:

1 volume		2 volumes		1 volume
CH_4	+	$2O_2$	=	CO_2+ $2H_2O$,

or a contraction in volume equal to twice the volume of the methane takes place. After burning the water rises at *E* to take the place of the burned-out gas and falls a corresponding distance in the tube *D*, *i.e.*, falls to a point opposite the scale that shows the percentage of methane originally in the sample.

The electric current for heating the platinum wire is derived from the small storage battery of a miner's electric cap lamp. The instrument is designed for connecting to this battery for test.

A determination can be made in two minutes with an accuracy of 0.10 per cent.

In Fig. 3 is shown an apparatus similar in principle except the electric current for heating the wire is derived from two small ever-ready dry cells *A* and *B* that are contained in the instrument itself. These cells cost about 7c. apiece and last for a minimum of 20 determinations. When used up they are as easily replaced as the dry cells in a flash light battery.

The essential steps in making a determination of methane in mine air are these:

1. One blows into the instrument, thereby filling the combustion space with water.
2. The water is allowed to fall, thus sucking a sample of mine air into the combustion space.
3. The electric current is turned on, thereby burning the methane.
4. The water at *D* falls showing the amount of methane in the sample.

An important point is this: After the gas has burned and the electric current has been turned off, the instrument is given a shake to force water into the combustion space and cool the hot gases. This brings the temperature of the gases to the temperature they had before burning and hastens the operation of making a test.

I am indebted to many members of the staff of the Bureau of Mines for assistance in perfecting these instruments. Dr. G. A. Hulett, Consulting Chemist, suggested the idea of building an apparatus of this principle, and Professor O. P. Hood, Chief Mechanical Engineer of the Bureau of Mines, rendered valuable service in perfecting the mechanical features.

W. E. Gibbs, New York, N. Y.—About two years ago the Bureau of Mines undertook the investigation of the causes of accidents, which were so often happening in the use of mine-rescue breathing apparatus.

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The Bureau commissioned me to undertake the investigation of the causes, an investigation of the apparatus most generally used, and the devising, designing, and construction of something which would overcome the difficulties that existed in the machines then used. I may say, incidentally, that I found the problem one of extreme difficulty. In the first place, there are so many elements in a device of that character that each has to be worked out as a separate problem, and the second and most important restriction was that a device of that kind has to be made so that it will not fail under any conditions that are conceivable, so far as human ingenuity and good workmanship can guard against them,

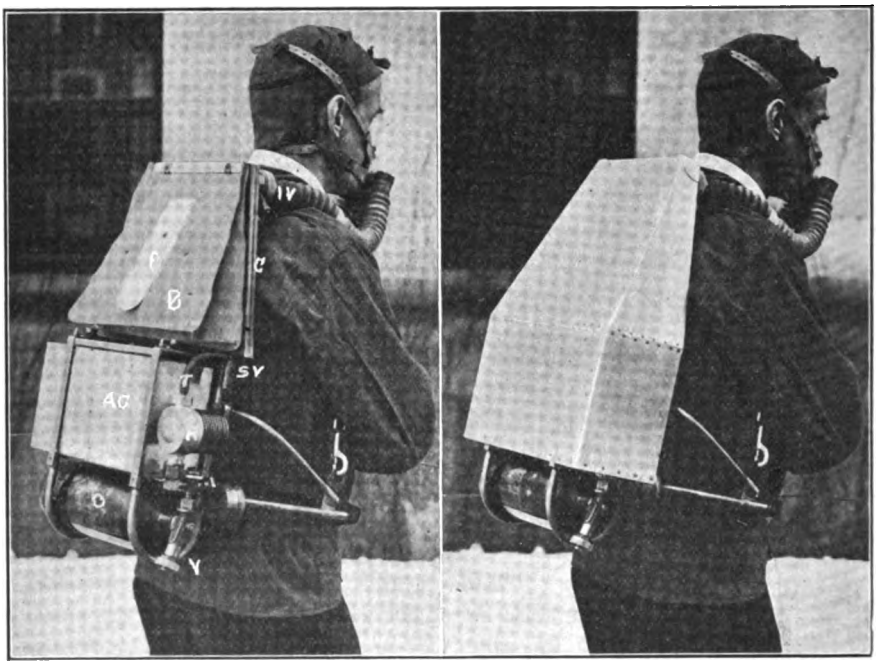

FIG. 1.—BREATHING APPARATUS WITH COVER REMOVED

FIG. 2.—BREATHING APPARATUS FOR USE IN MINE-RESCUE WORK.

as a failure almost always means that the man who wears the breathing apparatus dies.

I have brought with me the apparatus which represents the result of my labors as it exists at the present, and my assistant will put it on as the best way of illustrating what I want to tell you about it.

Now, in all mine breathing apparatus, as you doubtless know, there is a closed system in which a man breathes, a bag containing oxygen and purifying means whereby the carbon dioxide of the contained air is absorbed. After he has taken two or three breaths the amount of oxygen converted into carbon dioxide and removed by the purifying substance

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renders the total amount of oxygen in the whole apparatus inadequate to his needs, and the deficiency must constantly be supplied from a source of compressed oxygen carried in a steel bottle. All devices of this character are alike, as far as that goes. All forms of apparatus hitherto used have a breathing bag in front of a man's chest, and the bottles of oxygen and absorbing cartridge on his back. There is some advantage in that, because the weight of the apparatus on a man can be balanced nicely. To offset this, however, there are necessary connecting tubes passing under a man's arms from the rear to the front part of the apparatus which have to be big enough for easy breathing. Many loose ends were left hanging which were likely to catch on timbers or obstructions in the mines, and the apparatus was liable to be injured or the mouthpiece pulled away by such accidents.

One of the first ideas that occurred to me was that it would be safer to put the whole apparatus on a man's back, and to cover the exposed portions with a protecting shield made of aluminum for the sake of lightness. The cover on the apparatus has no function except that of protection.

One of the changes in the apparatus is that we are no longer using a helmet. For reasons that I will not go into, the helmet has been discarded by the Bureau of Mines because it is unsafe. We use a plain mouthpiece and nose-clip. These breathing tubes, which are long enough to go over the head when the apparatus is put on the back, rest on the shoulders and carry the air to the mouthpiece. When the wearer exhales through the exhalation tube, the air goes past a mica valve, which is very light, and rises at each breath to let the air into the copper box *C*. This has a partition near one end, which forms a flue, and carries the expired air down to one end of the absorbing can *AC*.

This can contains plates of caustic soda reinforced by wire-gauze screens so that as it wears away, the soda is supported by the metal and the surface remains the same. A surface of about 20 sq. ft. is maintained in this box. The plates are arranged vertically about a quarter of an inch apart with air spaces between them. The air goes down this passage into the absorbing can where the carbon dioxide is taken out by the caustic soda.

On the reverse side, the air passes up into a shallow box made of copper, *C*, blackened so that it may more readily radiate heat. Considerable heat is generated in the absorbing can by chemical action. The temperature in it rises to about 190° F., and the air issues with a maximum temperature of about 150° F. But by the time it gets up into this box and into the bag *B*, which contracts and expands as the user breathes, and through this long rubber tube into his mouth again, the temperature sinks to about 98°. It seems to be unnecessary to provide any artificial

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means for cooling, by the use of sodium sulphate for instance, as is being done in some apparatus.

A special feature of this apparatus is this reducing valve *R* which we hear chattering. That chattering is a little vibration of the valve against the seat during the admission of oxygen. That bothered me at first, but all the men in the Bureau of Mines warned me not to do anything with it, as it was so comforting. The men said that as long as the valve chatters they know that the oxygen is flowing.

The reducing valve takes oxygen from the bottle *O* at a pressure of 2,250 lb. per square inch, or 150 atmospheres. Instead of a rubber diaphragm for regulation, a bronze bellows is used, such as is employed by the heating companies, for steam-heat regulation, to operate dampers and things of that kind. This expanding bellows works a toggle-joint that closes the valve. The gas that issues from the valve has a pressure of 2 lb. per square inch, a reduction of 1 to 1,000.

The valve proved so perfect in operation that the first one I made has been used for several months, intermittently, many hours during that time, without even taking it apart. When it was finally opened for examination, it was again put together, after inspection, and worked as well as ever. It has been working for perhaps a year and a half without any sign of deterioration or any change in adjustment. That will answer the questions which will naturally arise as to its permanency.

The oxygen passing through the valve to replenish the system goes through this tube *T* and up a small copper tube within this cooling can *C* to the admission valve which is directly in the center of the opening into the breathing bag. There the gas is absolutely checked. When the man breathes, the bag expands and contracts maybe several times before any further oxygen is required. When, however, the supply is insufficient, the bag collapses far enough to permit the weight inside the bag to open the admission valve and more oxygen enters.

E. M. CHANCE, Wilkes-Barre, Pa.—Is there a check valve on the exhalation?

W. E. GIBBS.—There is a circulating valve.

E. M. CHANCE.—That apparatus does not work on the injector principle; the man circulates the air through the apparatus with his lungs. Is the whole apparatus under positive pressure?

W. E. GIBBS.—There is always a little positive pressure in it and no injector.

E. M. CHANCE.—What is the capacity of the cylinder?

W. E. GIBBS.—Two-liter cylinder charged at 150 atmospheres. Under the ordinary conditions, with the older machines, with the oxygen current flowing constantly at 2 liters per minute, the charge would last

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for 2 hr., but we find the average consumption is much less in this machine, so that for two hours 90 atmospheres suffices. It has gone 2.5 hr. on 90 atmospheres, under test conditions.

I omitted to say that there is a pressure gage, which, instead of having a flexible tube carried around in the usual way to the front so that the user can see it, it is fixed to the side of the apparatus and is read by touch. The danger that the flexible tube may be caught in something and injured is thus avoided. Moreover, inside of the pressure gage is a little alarm, a movement out of an alarm clock, and if the pressure falls to 30 atmospheres the alarm goes off and gives the user notice. The alarm does not, under normal conditions, go off unless the apparatus is worn more than 2 hr., but if there were a leak in the system it would very soon go, and the reserve would presumably give the wearer time to get to a place of safety.

T. M. CHANCE.—Can you carry a man on your back with that apparatus?

W. E. GIBBS.—I think not.

E. M. CHANCE.—I have had eight years' experience with breathing apparatus in connection with the Philadelphia Iron Co., and during that time we fought about 15 mine fires, and I had about 1,700 men under my direction. The last fire extended over 41 days.

The usual types of apparatus, with the exception of the Fleuss, are adjusted so that 2 liters of oxygen are the limit. Under the ordinary conditions of a man standing and walking as usual, that is adequate, but under the severe mental stress, high temperature, and excessive work that is necessary to be done in recovery work in mine fires, that supply of oxygen is inadequate, and many if not all the fatal accidents which have occurred in rescue apparatus I attribute to the fact that the men were using oxygen faster than they could get it; the symptoms of suffocation being frightful, the men go down with the apparatus and tear it off.

It seems to me that in this apparatus that source of danger is removed, and the men receive oxygen as required. I think they will require as much as 100 liters of air per minute; they can get it with this apparatus, and you will find, under practical conditions, that there are men who have used up the air in that apparatus within an hour, and it is right that they should. Give a man an adequate supply of oxygen for an hour, and his place after that time is not with the apparatus on his back, but out on the reserve resting. That, in my mind, is a fundamental feature which is most admirable in safeguarding the men and permitting them to do heavier work.

I have found two things essential to satisfactory rescue apparatus work, and one of them is the most stringent inspection of the apparatus

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at most frequently recurring periods. If the apparatus has been inspected and placed in the corner for a period of 15 min., there really is no reason for believing that at the expiration of those 15 min. the apparatus is in perfect condition. It must be examined frequently, and these frequent inspections are absolutely essential, and in addition to that, the workers who wear the breathing apparatus must be under the control of a teacher who will keep the working speed of the crew at the rate of the slowest, weakest, and most frightened man. You will find that under the conditions of actual service, in the use of all rescue apparatus, and under actual conditions of fire-fighting and recovery work, the men are frightened. Their metabolism is speeded up so that the quantity of air they require is exorbitant, and unless you can keep their speed of work down to the lowest possible limit, there will be fatal accidents. I have had conditions where we had to have the men lie down on their backs every 10 min., and every 10 ft., sometimes, and when we could keep them down to that slow work we had no trouble. We have had them fall in 15 and 20 ft., and knock the apparatus from their heads, but that is usually in the case of the inefficient helmet.

T. M. CHANCE.—I think your apparatus promises to be most satisfactory. What is the weight?

W. E. GIBBS.—It is 30 lb.; about 10 lb. lighter than the others.

T. M. CHANCE.—As to the cartridge?

W. E. GIBBS.—We have decided for the present to throw away the cartridge after it is used. It does not seem safe to trust to the re-use of the cartridge.

T. M. CHANCE.—Another point, do you know the extent to which you were condensing nitrogen? Now, there is a certain volume of oxygen in that solution, about 10 cu. ft. Would you say that there was about 5 per cent. of nitrogen?

W. E. GIBBS.—Before answering that question I will say, as a matter of fact, that 5 per cent. is rather an excessive amount. The operator, in starting the apparatus, is instructed to take two or three deep breaths and exhale through the nose, in order to get as rich an oxygen mixture as possible, and this reduces the nitrogen. The danger of excessive nitrogen is great, because it is insidious; there is no warning; a man loses consciousness much as when poisoned by carbon monoxide. By starting with an atmosphere of 90 per cent. of oxygen, you experience no inconvenience whatever, there is no more oxygen absorbed than the normal quantity, and if there happens to be 50 or 75 per cent. of nitrogen in the machine by the time you finish using it, you are still on the safe side.

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The Effect of Aeration and "Watering Out" on the Sulphur Content of Coke

BY J. R. CAMPBELL,* B. S., M. S., SCOTTDALE, PA.

(New York Meeting, February, 1916)

In order to discuss the subject intelligently, it will be necessary to touch briefly on the forms in which sulphur is supposed to exist in coking coal to be carbonized in beehive or byproduct ovens.

Sulphur is known to exist in coal as sulphides and sulphates, as can be determined experimentally. Then there is the so-called organic sulphur, *i.e.*, sulphur in combination with the carbon, hydrogen, and oxygen of the coal, about which much has been written, but nothing definitely proven in an experimental way. In fact, so far as we know, there never has been developed a satisfactory and conclusive method in the laboratory for the direct determination of organic sulphur.

For most coking coals, it can be safely assumed that the preponderance of sulphur is in the form of pyrite (FeS_2). When exposed to comparatively low temperatures during the coking process, it loses its sulphur according to the following chemical reaction: $7FeS_2 = Fe_7S_8 + 6S$. It will be seen, therefore, that six out of the 14 atoms of sulphur (or 42.8 per cent.) are expelled, or volatilized. Fe_7S_8 remains in the coke as pyrrhotite, or magnetic sulphide of iron; this accounts for the highly magnetic properties of the powdered coke. A more commonly accepted reaction is $FeS_2 = FeS + S$, in which the straight sulphide of iron is produced with 50 per cent. volatilization of sulphur.

In the beehive methods of coke making, air is introduced in sufficient amounts to carry on the distilling and the coking processes, and the sulphur is oxidized along with the other volatile products: First, into sulphur dioxide (SO_2), known by the pungent and suffocating odor emitted from the trunnel head; second, into sulphuric anhydride (SO_3); and, finally, into sulphuric acid, as it comes into contact with the air and moisture. In a properly regulated draft on a beehive oven, there is never complete combustion of the gases. In other words, the coking process should be carried on in a reducing atmosphere, and a low-grade producer gas kept issuing from the trunnel head. For coals of about 30 per cent. volatile matter, the ratio of air to gas is 3½ to 1. In complete combustion, the ratio is 6 to 1, producing an extremely high temperature

*Chief Chemist, H. C. Frick Coke Co.

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in the crown of the oven (3,500°F.) more than enough to cause fusion of the refractories used. A good coking temperature is 2,500°F. in the crown of the oven.

It follows then that the sulphur remaining in the coking mass as iron sulphide cannot in any way be affected by the aeration or draft on the oven. Among beehive coke-oven operators there used to be a saying, "the hotter the oven, the more sulphur burned out." In view of the foregoing, this probably never was true unless the aeration was carried to complete combustion of not only the volatile gases but the fixed carbon itself, in which case the iron sulphide (FeS) would, of course, be oxidized to Fe_2O_3, and the sulphur liberated, as usual. This condition would not be productive of good results, as the percentage of coke yield would be abnormally low and the "ashes," or "braize," correspondingly high. This is evidence that so far as the pyritic sulphur is concerned, any sulphur that is volatile at all is so at comparatively low temperatures through the agencies of distillation, and no practical method of superaeration will help in the least in the further elimination of sulphur.

On the other hand, overheating promotes certain chemical reactions in which sulphur forms compounds with other bodies on which heat has no effect. If the coking process proceeds too quickly, or if the heat is irregular, the desulphurization of the coke is apt to be less complete. In these matters, much depends upon the skill and judgment of the burner.

In the byproduct oven, which is essentially a true distillation process, the heat being applied externally, and no air supplied to the oven itself, the sulphur is distilled from the pyrite as atomic sulphur, as previously explained, but, apparently, it afterward combines with the hydrogen of the coal gas, forming sulphuretted hydrogen (H_2S), for it is necessary to remove this very objectionable gas before artificial gas can be used for domestic purposes. This is accomplished by the use of hydrated oxide of iron, a byproduct now obtainable from mine-water purification.

Here again appears the fallacy of superaerating beehive ovens to eliminate sulphur, for in byproduct ovens in which no air is admitted there is just as much desulphurization of the coke as in the beehive process; in fact, there is more desulphurization when the matter of increased yield is considered. For example: Given a yield of 70 per cent. coke by the beehive process and 75 per cent. by the byproduct process from coal carrying 1 per cent. sulphur, the respective problems are as follows:

	Beehive Oven, Per Cent.	Byproduct Oven, Per Cent.
Sulphur in coal	1.000	1.000
Sulphur volatilized	0.428	0.428
Sulphur remaining	0.70)0.572	0.75)0.572
Sulphur in coke	0.817	0.763

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Thus the practical volatilization of sulphur from the beehive process is only 18.3 per cent., while in the byproduct it is 23.7 per cent. In our own beehive practice we have always used 20 per cent. as the amount of sulphur eliminated from all sources, figuring from coal to coke, this being the result of numerous laboratory and practical tests on the Connelsville coal.

Attention has been directed in the preceding paragraphs to the sulphur as pyrite. Let us look into the other forms that may be present in addition to the sulphide. Any organic sulphur present remains, for the most part, in the coke. It is readily seen how it would be unaffected by aeration except in event of total destruction of the carbon itself, with which the organic sulphur is supposed to be combined.

Any sulphates present, such as $CaSO_4$, would also be unaffected by aeration or superaeration in the beehive process; no amount of "airing" would help to eliminate the sulphur. We know of cases in which the sulphur in the resultant coke was as high as, or even higher than, in the coal, from which it must be concluded that much of the sulphur was present in the coal as sulphate, or the so-called organic sulphur; or else the ash of the coal was rich in iron, lime and magnesia, for there is a dictum that it is never possible to produce a low-sulphur coke from the coal the ash of which is high in these elements. Superaeration cannot and will not ameliorate these conditions.

The Effect of Quenching on Sulphur

Perhaps everyone is familiar with both the beehive and byproduct oven practice in this country, quenching the coke by copious amounts of water. In the former case, it is customary to "water out" in the oven itself, while in the latter it is done externally by means of suitable quenching stations. In beehive practice, it takes approximately 1,000 gal. of water to quench a 5-ton charge of coke, consuming about ¾ hr., either by hand watering with a hose, or with a sprinkler of the Stauft type, which is placed in the oven on top of the coke. The byproduct practice is much quicker, but the external watering produces darker coke.

Theoretically, it would seem possible to remove considerable sulphur by the quenching process. We have already indicated how iron sulphide (FeS) is formed in the coking process. The action of the water on the sulphide is as follows: $FeS + H_2O = FeO + H_2S$, in which it is assumed that the coke is quenched externally. The odor of sulphuretted hydrogen is easily detected (but it will be appreciated that a little of this gas goes a long way), and the color of the coke, wherein rust spots appear, shows the presence of iron oxide. The practical coke burner is always suspicious of what he terms "rusty coke," as it invariably is an indication of high-sulphur coke. The same holds true

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in quenching in the oven itself, as in beehive practice. Sulphuretted hydrogen (H_2S) is evolved, but, at the same time, the water as steam is decomposed by the carbon, $H_2O + C = CO + H_2$, which fact probably accounts for a part of the large percentage of carbon monoxide and hydrogen gas found in the gas from trunnels of beehive ovens, the water being formed from the combustion of the gases during the coking processes, which is virtually a low-grade producer gas of the following composition:

	By Volume, Per Cent.
CO_2	3.0
CO	9.0
H_2	11.0
CH_4	0.3
N_2	76.7

The desulphurization of coke by water cannot be complete. The coke mass cools too quickly and its very structure prevents the rapid penetration of the water thrown on it, especially in the denser varieties. As the temperature is lowered the reaction involved becomes too slow to be of practical benefit. Data as to the exact amount of sulphur eliminated in this way are rather scarce, but our own experience, based on the general laws of volatilization set forth elsewhere, leads us to the conclusion that only an infinitesimal percentage of sulphur is thrown off during the quenching process. Laboratory and practical tests, in which the coke has been allowed to cool naturally, show but little difference in the sulphur content from those in which the coke has been quenched with water. Certainly there could be only a few hundredths of 1 per cent. in favor of the water-quenched coke, at the most, in a coke averaging 1 per cent. sulphur.

The prime object of the use of water is to lower the temperature of the coke for handling, not for desulphurization, and the quenching process should be so regarded. However, we are of the opinion that quenching by the byproduct practice will eliminate more sulphur than by the beehive method, which may partially explain the greater total volatilization of sulphur in the former, elsewhere intimated.

It may not be generally known, in connection with the subject, that addition of muriatic acid (HCl) to the water greatly facilitates the removal of sulphur during the quenching process. The action of this acid on iron sulphide is positive at all temperatures, thus, $FeS + 2HCl = FeCl_2 + H_2S$. There was, of course, a time when the cost prohibited the use of this method, but with the depletion of our low-sulphur coking coals this may, in the near future, be a factor in the elimination of sulphur.

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Economies in a Small Coal Mine

BY HERBERT A. EVEREST,* B. S., E. M., M. E., OKLAHOMA CITY, OKLA.

(New York Meeting, February, 1916)

THE idea of economical production is usually associated with large operations, tonnages, and mines, with even larger capital behind them. Nevertheless many small mines operate in the shadow of large competitors and make a good showing on the capital invested despite larger overhead expenses.

For the purpose of this discussion I will divide the cost of production into classes, and specify opposite each the approximate percentage expended thereon:

1. Labor, including miners and company men; 60 to 75 per cent.
2. Development, including all necessary yardage, room turning, crosscuts, etc.; 7 to 12 per cent.
3. Deadwork, covering payment to miners for handling falls, draw-slate, or faults, rock work, water, and in general, all nonproductive labor; 3 to 7 per cent.
4. Supplies: mine timbers, oil, brattice material, lumber, cement, and repairs to equipment; 2 to 6 per cent.
5. Expense: management, selling, office, taxes, etc.; 1 to 5 per cent.
6. Depreciation of coal reserves and royalties; 6 to 10 per cent.
7. Depreciation of equipment and interest on capital invested; 1 to 3 per cent.
8. Fuel; ½ to 1½ per cent.

The labor cost for a small mine is relatively much lower than for a large mine, particularly the sum paid to company men. Idle-pay expense is cut to a minimum; the labor item being larger than all the others together, a saving in labor makes a decided showing on the total cost.

The development charges in small and large mines are about the same. There should be a small showing on the deadwork item in favor of the small mine. The unit cost for supplies is usually small in the small mine. The operator of the small mine finds, owing to a limited tonnage, that the expenses of management, selling, etc., are abnormally high.

The item, depreciation of coal reserves and royalties, is usually about

* Superintendent, The Hazelton Coal Co.

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equal for both small and large mines; while the small mine is favored with regard to depreciation of equipment and interest on capital invested. If the mine is a shallow one, the fuel account is also in favor of the small mine—as a rule, not because of any great effort on the operator's part.

In the small mine the principal savings are in labor, supplies, equipment, and capital, because the working places are closer together, thus concentrating the working force. Fewer mine cars, less mine trackage, fewer entries and air courses are required and need be maintained, and the surface equipment is cheaper.

Concentration of the working force brings the mine labor under close supervision; and where fewer men are employed, all work to better advantage. The movement of cars is rapid and the miners load a good turn, producing a greater tonnage per unit of area opened. Concentration and close supervision result in the operation of the mine on a very small capital investment per ton of production, the charge for interest and depreciation of equipment being proportionately low.

Under close supervision supplies are not wasted, misused or lost, in many instances materials being used over and over again. Entries are usually worked out during the life of one set of timbers so that the expense of retimbering is obviated.

The operator of a small mine has a keen regard for the inroad on his profits that even one extra man on the payroll would produce. He realizes that much valuable time is wasted in the mine—some of it due to laziness, but more to misdirected energies and lack of or improper planning, so that the men are obliged to wait for material, tools, or cars. The chances are, in a small mine, that if a place is to be timbered, a fall is to be cleared up, or a room switch to be put down, all of the material is on the spot before the man arrives to do the work. The different working places are close together, therefore the workman does not waste much time in going from one place to another. In this way one tracklayer can do the work of two men who move long distances between jobs, and must look over the place, order their own material, and probably wait for its arrival.

At the rate of $3 per day, and working 200 days in a year, a man receives $600 per year; in 10 years he has received $6,000. During that time he has been a liability, because of the possibility, through carelessness on his or some other miner's part, of an accident or fatal injury. Any mechanical device, which might supplant a man, the first cost and upkeep of which would be less than $6,000 in 10 years, would be a good investment. The mechanical device will not lay off or quit; this precludes the necessity for training new men on the job, often a costly operation. The machine does not come under any workman's compensation act; it does not strike nor get out of repair as often as the human machine (provided it is properly designed and cared for).

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Some of the comparisons made are unfair, but inasmuch as the small mine usually has a limited acreage and frequently shallow coal, it follows that the mine will never have many long entries, that the haul will be short and the work, while it lasts, will be concentrated.

Reverting to basic principles, the economies in a small coal mine result almost solely from close concentration of the working places and a still closer supervision of the labor employed.

Discussion

Newell G. Alford, Earlington, Ky. (communication to the Secretary*).—Mr. Everest's statement concerning the economies in a small coal mine has interested me very much. The majority of discussions on coal-mine economy omit the small operator and for this reason Mr. Everest's paper should lead to productive thought.

I agree that close concentration of working places and constructive supervision of labor are two items of salvation to the small operator. It is also true that small mines often yield as much profit per capital invested as their larger neighbors.

Mr. Everest also says that the small operator has so keen a regard for the inroad on his profits that even one extra day hand is an extravagance not tolerated, and this is rightly so. But, does not this earmark of the small operator in many cases offer a clue to his vital weakness?

A credit balance is the acid test of success in any enterprise. With the small operator who owns practically all the capital stock this is particularly significant. He never takes his eyes off the market price and at the same time strains every nerve to cut down his payroll. In several cases I have noted that the resultant close supervision of labor was in fact too close. The "stitch in time" was not applied to the repair of this or that, with a decided increase to the cost of the work when it had to be done at the expense of cutting off the run when orders were on the books.

By nature the small operator is economical. Therefore, he salvages and re-uses material, but maximum overall efficiency in his plant is often the factor which he heeds not. He seldom partakes of the exchange of thought on this line of endeavor because the first cost thereof adds directly to his expense. Unless his previous training has been so directed, and this is rare, he fails to sense the possibilities of the systematic investigation which begets a self-sustaining efficiency. In short, the point I have in mind is that over-concentration diminishes the scope of vision and often leads to nearsightedness.

I am of the opinion that the average small operator could add much to his final profits by a systematic cutting of production costs through the application of organized effort in this direction.

* Received Feb. 8, 1916.

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Brown-Coal Mining in Germany

BY GEORGE J. YOUNG,* B. S., MINNEAPOLIS, MINN.

(New York Meeting, February, 1916)

DURING the spring of 1910 I visited a number of open-pit brown-coal mines and underground workings in the vicinity of Halle, Halberstadt, Leipsic, Cologne and Bonn. The notes which I took and the observations which I made at the time have been condensed and make up the principal part of this paper. Certain supplementary information has been taken from several sources which are given in the paper.

The mining of brown coal is one of the important industries of Germany. The coal varies in color from dark brown to almost black and is soft like loam. Pieces of wood and strips of bark and even parts of tree trunks are frequently encountered in the excavation although the coal for the most part consists of a pulverulent, more or less matted mass of small particles of vegetal matter. It is a product between peat and lignitic coal. An analysis, taken from *Coal Resources of the World* shows the approximate composition of the material: Moisture, 53.73 per cent.; ash, 4.98; fixed carbon, 18.08; volatile carbon, 23.31 per cent. The principal characteristic is the amount of moisture. This seldom runs less than 48 per cent. The ash is usually low and ranges from 5 to 10 per cent. The sulphur content is usually less than 0.5 per cent. The heating effect of the fuel ranges from 2,000 to 2,500 calories.

The commercial deposits are confined to the Tertiary period although

	Reserves in Millions of Tons	
	Actual	Probable and Possible
Prussia and North German States.....	6,069	3,676
Saxony.....	3,000	.....
Bavaria.....	75	293
Hesse.....	170	99
Total.....	9,314	4,068

* Professor of Mining, University of Minnesota.

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some deposits, not of commercial importance, occur in the Pleistocene epoch. In Saxony and Thuringia, which produce about 47 per cent. of the German output, the deposits are found in the Eocene-Oligocene division of the Tertiary. Most of the other deposits are found in the Miocene division. The distribution in Germany is given in the preceding table.[1]

The important centers about which brown coal is mined are Bonn, Cologne, Halberstadt, Halle, Leipsic, Semphenburg and Brunswick.

The deposits present a great variety in shape and size. They range from relatively thin beds, 9 to 30 ft. thick, up to great basin deposits

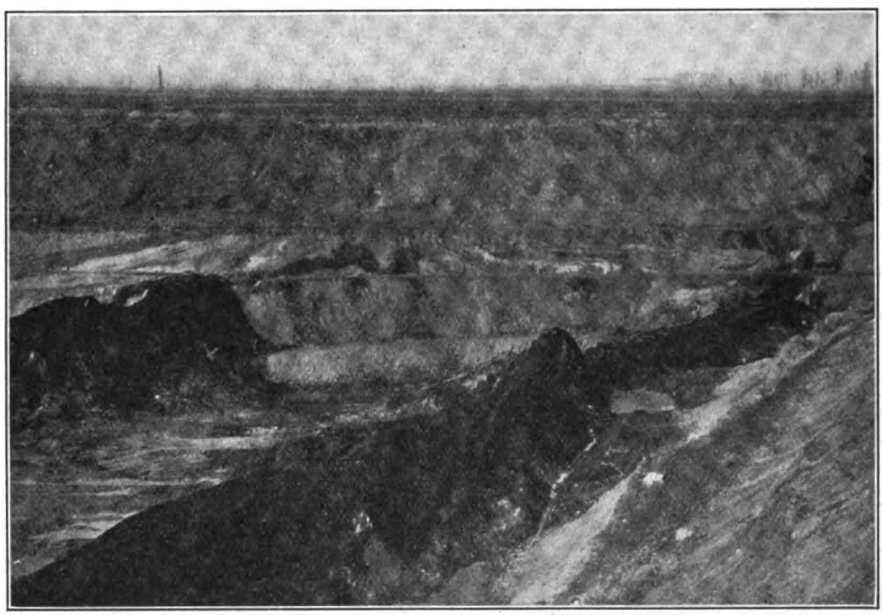

FIG. 1.—BROWN-COAL PIT NEAR BORNA.

100 to 300 ft. in thickness. They are contained in clays, sands, sandstones, marls; in some instances limestone beds are intercalated. They are more or less erratic in distribution and extent. Many of them are covered by glacial drift consisting of fine sand and light gravel. This may be only a thin layer from 20 to 25 ft. thick, or may range from 75 to 300 ft. in thickness. Faulting was not conspicuous in the pits visited with the exception of one in the vicinity of Borna, south of Leipsic. The latter deposit was trough-shaped, the main axis of the trough being cut by four transverse faults. Fig. 1 shows the edges of two of the blocks. The non-conformable glacial drift above the inclosing formations and the flat topography of the country south of Leipsic are also shown.

[1] From *Coal Resources of the World*, vol. i, p. 89.

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Open-Pit Methods

Wherever the thickness of the overburden permits, the brown coal is won by stripping the overburden and excavating the coal by modifications of the "milling method." In most of the pits visited the overburden consisted of glacial drift, remarkably free from boulders, composed principally of sand and to a less extent of clay. The flat nature of the topography requires the excavation of the pit below the general level of the surface and greatly facilitates the placing of the surface plant and the arrangement of the tracks for the handling of the spoil.

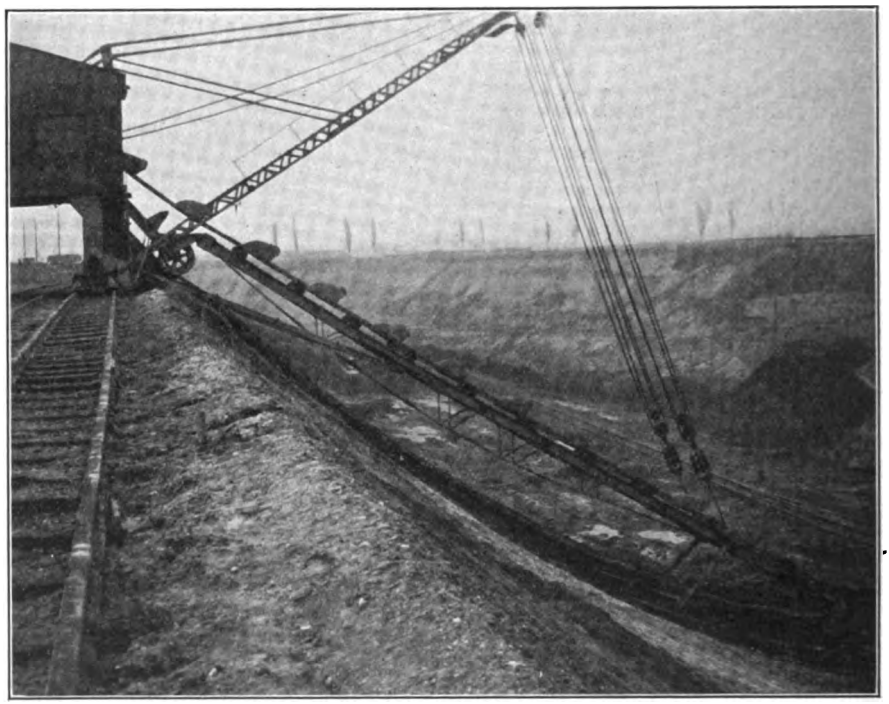

FIG. 2.—CONTINUOUS-BUCKET EXCAVATOR FOR REMOVING OVERBURDEN.

The overburden is excavated by a continuous-bucket excavator of a type practically unknown in the United States (Fig. 2). The Parsons trench excavator is quite similar in principle. The buckets of the excavator are attached to a pair of heavy chains which are supported by two sprocket wheels and rollers, the latter in turn being supported by a steel frame or ladder. The ladder is pivoted at the upper end so it can be raised and lowered through an angular range from horizontal to 45° below horizontal. The arrangement is somewhat similar to the ladder and bucket run of a continuous-bucket dredge. The construction

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is much lighter and the chain of buckets is operated in the reverse direction as compared with the dredge, that is, the empty buckets, mouth down, descend on the upper side of the ladder while the loaded buckets are drawn up on the lower side. As the buckets pass around the upper sprocket wheel the contents are dropped into either a hopper or a metal chute. The discharge hopper or chute is set at a sufficient height to allow the spoil cars to pass beneath. The digging ladder and driving mechanism are integral with a platform supported by trucks which run on a double pair of rails. The distance between the two sets of tracks is sufficient to admit of the placing of another track upon which the spoil cars may run beneath the loading platform. The excavator can be driven by its own power over a train of spoil cars, filling the cars as it goes, or the train of cars can be shunted into position by a light locomotive.

While there are many modifications of the excavator above described, I only saw two types in operation. One is called the *tiefbagger* (deep excavator) used in making a cut below the level of the track upon which the machine operates; the other is the *hochbagger* (high excavator), used in making a cut above the track level of the machine. The machines are made for different capacities and depths of digging. The following table will give the range of sizes and capacities of the machines of this type manufactured by the Lübecker Machinenbau-Gesellschaft:

TYPE

	B	*A*	*C*	*F*	*L*
Capacity in 10 hr. digging time, approximate cubic yards:					
Easy soil	2,400	1,800	900	400	220
Medium soil	2,000	1,500	700	300	170
Hard soil	1,600	1,200	500	200	120
Deepest cut, feet	45	30	24	15	15
Horsepower (approximate)	90	50	38	16	12
Coal consumption in working time, pounds	3,860	3,200	1,100	880	660
Capacity of buckets, cubic feet	8.5	6.4	3.5	1.8	1.25
Cost of machine F.o.b. Lübeck From	$11,250	$8,750	$6,250	$4,500	$2,750
Cost of machine F.o.b. Lübeck To	13,000	10,250	7,500	5,500	3,500
Shipping weight, metric tons	70	48	34	22	12
Number of men required	2 to 3	2 to 3	2 to 3	1 to 2	1

Approximate cubic yards represent cubic meters in the original table; table taken from H. Bansen's *Die Bergwerksmachinen*, vol. ii, p. 35.

The excavator employed in the pits visited makes a cut about 25 ft. deep and leaves a face sloped to 45° or less as conditions require. In operating, tracks are laid along the line of the cut; when the cut has been completed the tracks are shifted back and a new cut taken. Spoil

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is handled in small cars hauled by light locomotives. The spoil dumps are built up on the surface or backfilled in the pit. They present no particularly novel features.

From one to three cuts or benches are sometimes necessary in stripping a deposit. As soon as the coal is exposed a steep incline is excavated in the coal and extended until a working face from 50 to 100 ft. in vertical height is obtained. From the floor of the pit thus established, drifts of small cross-section are driven at intervals of 50 ft. into the base of the bench. At intervals of 25 ft. along the course of a drift, chutes are constructed to the surface. At the mouth of each chute a crater is started

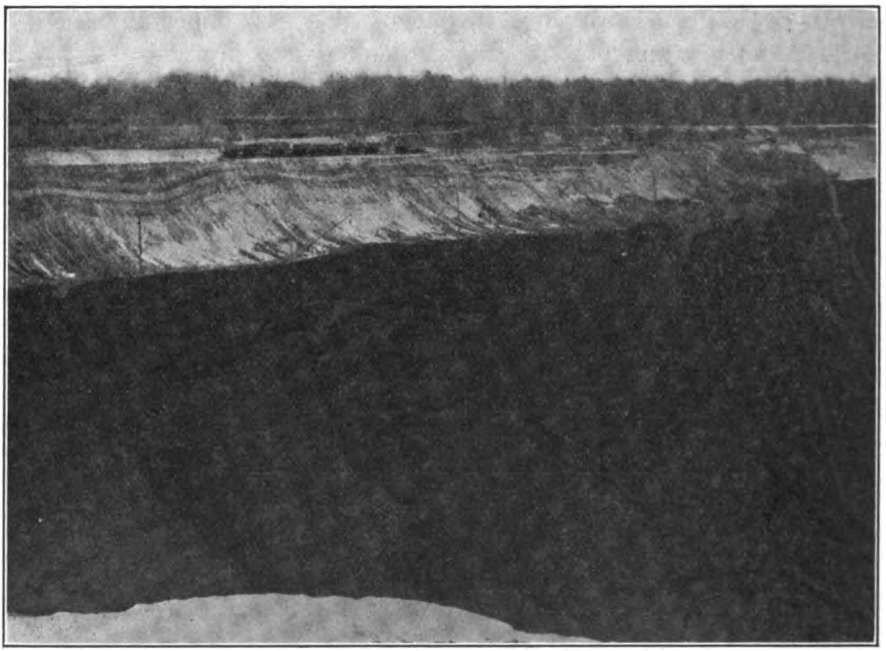

FIG. 3.—OPEN-PIT MINING SHOWING STEEP "CRATERS."

and the coal worked into the chute. The coal is excavated by hand, the worker using a single-pointed pick and cutting the coal from the bottom of the crater upward. Footholds are cut in the sides of the crater, and, where extremely steep, ropes are used to prevent falls. The steepness of the craters is well illustrated in Fig. 3. The crater is worked out until the sides reach a slope on which the coal fails to slide. Drifts extend from the open floor of the pit radially or transversely along an axis of the pit. The ribs which are left between the drifts are excavated from the floor of the pit. By using a movable chute mouth, constructed of wooden planks and braced by inclined diagonal struts against a vertical face of coal, a large proportion of the coal in the rib can be won without shoveling.

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The stump of the rib and the coal excavated in squaring up the face preparatory to moving the chutes must be shoveled. In excavating thin portions of a bed the movable chute is used instead of the drifts and the mill holes.

Drifts are constructed as small as possible and the back is arched so that they will be as near self-supporting as possible. Little timber is used and that principally at the point where the vertical chutes connect. The cars used are from 28 to 30 in. wide and 3½ ft. high above the rail. The car is round bottomed, 21 cu. ft. in capacity, constructed of steel; it is dumped by means of dumping cradles. The chute gates are simple

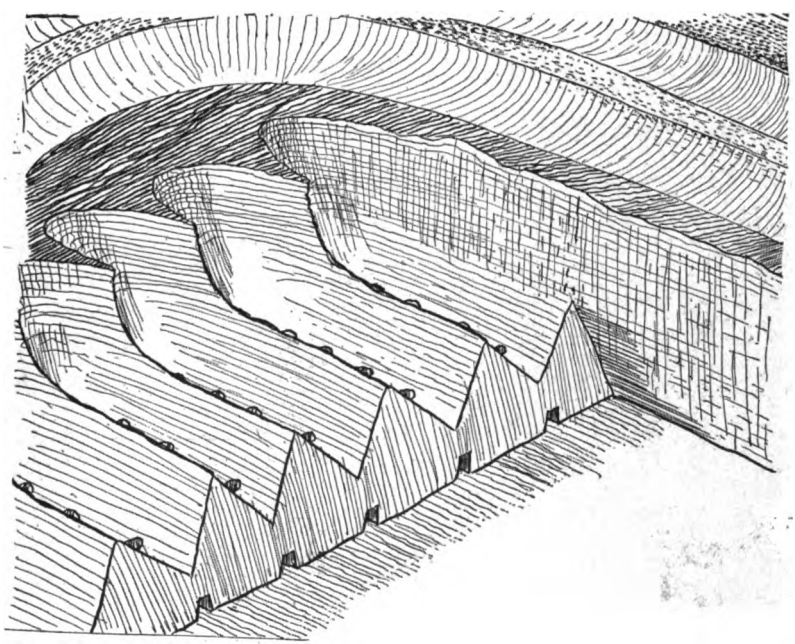

FIG. 4.—APPEARANCE OF A PIT IN WHICH A MECHANICAL CUTTER WAS USED TO MINE AND LOAD THE COAL.

in design, consisting of two or three boards held by cleats on the side boards of the chute.

The general layout for opening up the pit depends upon the shape and size of the deposit. I noted two general systems used on the larger and thicker deposits. In one the initial cut is started at the foot of the main incline and is extended parallel with and on the longer axis of the deposit. This can be done by starting three or four parallel drifts from the foot of the incline and extending them parallel with this axis. Mill holes are developed and follow up the drifting. The ribs are taken out and the floor of the pit cleared. At right angles to the first cut, parallel crosscuts are extended at intervals of 50 ft. and as rapidly as the flanking walls of

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the initial cut permit. Craters are then started in the flanking walls. In the other system the initial cut is made transversely to the main axis. From the pit floor thus developed, drifts are driven parallel to the main axis of the deposit. The line of advance of the craters is parallel to the main axis. In one working near Bonn this method of opening the pit was observed. The coal in the ribs was won by means of a mechanical cutter which not only cut the coal from a face which was continuous across the pit but also loaded it into cars. The appearance of this pit and the face developed by the mechanical cutter is shown in Fig. 4. The

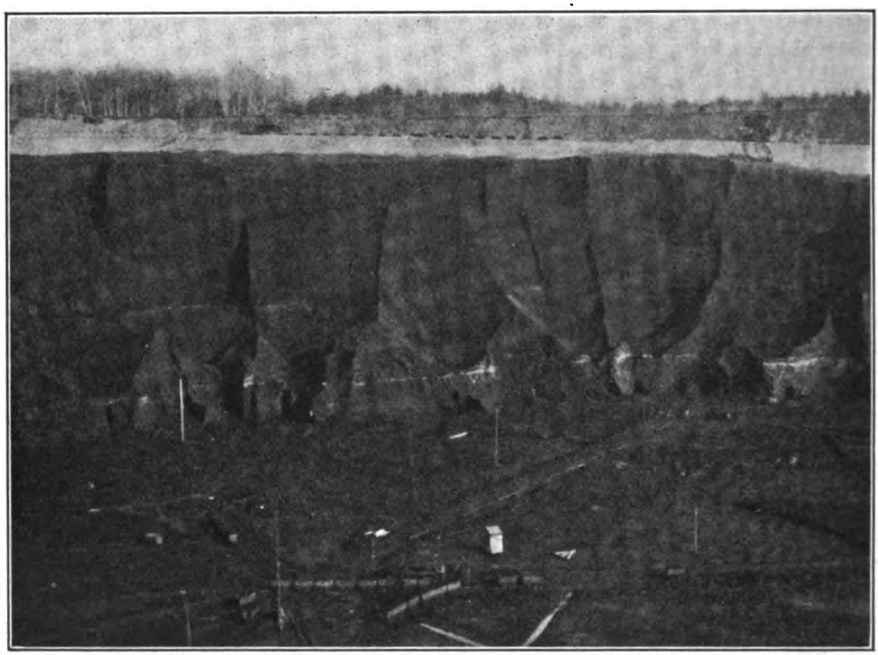

FIG. 5.—PIT IN OPERATION SHOWING SYSTEM OF TRACKS AND THE "MILLING" METHOD IN USE.

sequence of the stripping influences to a certain extent the method used in laying out the pit and usually favors the method first described.

The transportation problem is of interest as it involves methods which have probably been developed in this form of mining. The problem consists in the handling of a large number of small units or cars which are loaded at a number of points and which must be moved to a common dumping point. The cars are hand-trammed from the drifts and crosscuts over relatively short lengths of track which may converge at a common point or which may be parallel and terminate in a common track transverse to the drifts. A double track equipped with chain haulage serves the secondary tracks or the common points. The chain, consisting of

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4-in. links, is driven by a sprocket wheel and gears at the end of the run. Motor drives are customary. A plate in which is cut a V-shaped slot projects upward from the end of each car and engages with the links of the chain. The chain is supported by the cars which are spaced at 20 to 40-ft. intervals, but is carried around corners on guide sheaves. At each turn the track is graded so that the cars gain speed on the chain and disengage (the chain is elevated sufficiently at the turns to accomplish this), gravitating around the turn where they are picked up by the sag of the chain and hauled along the next tangent. One or more runs may be necessary in a given pit. Loads are taken out on one track and the empties returned on another. The incline giving access to the pit is served by a chain also. The system is almost automatic and delivers the loads to the tipple and returns the empties with but little manual assistance

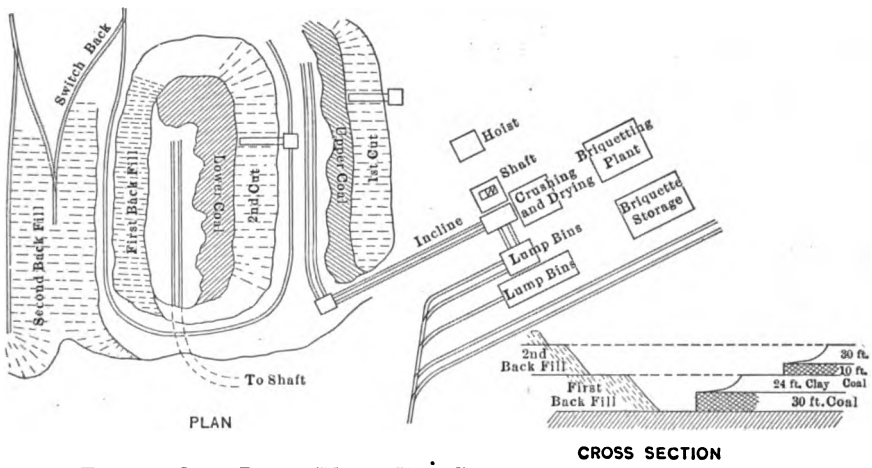

FIG. 6.—OPEN-PIT IN WHICH TWO COAL BEDS ARE BEING WORKED.

except at the points where the cars are attached and the empties removed. The *kettenban* (chain railroad) works on the same principle as the continuous-rope haulage system. The system as a whole made a favorable impression upon me.

In one of the smaller pits near Halle the upper portion of the overburden, consisting of sand and soil, was stripped by hydraulicking and backfilled into the lower part of the pit. Retaining dams were put in on the toe of the spoil slopes, the sluices discharging back of them. The remainder of the overburden, consisting of moderately compact shales and clays, was removed in two benches by undercutting, caving, and shoveling by hand into cars. The cars were hauled in trains by horses to the backfill.

Drainage in the pits is a comparatively simple problem. Electrically driven pumps connect with a sump in the lowest part of the pit and are operated when necessary.

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Fig. 5 shows a pit in operation and illustrates the system of tracks and the milling method in use. Fig. 6 is a sketch plan of an open pit in which two beds were being worked.

Underground Methods

The conditions which limit the underground methods are: Relatively low selling price of the coal, extensive thick deposits, soft spongy material,

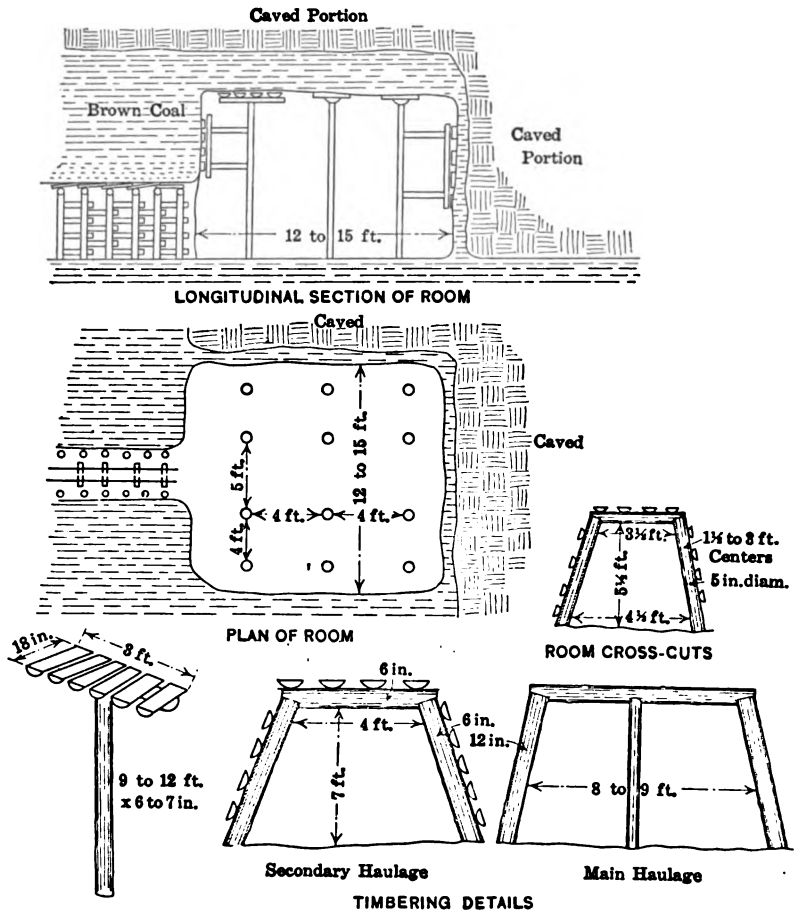

FIG. 7.—UNDERGROUND MINING SYSTEM.

a cover of sand and soft sedimentaries, depths ranging from 200 to 300 ft. and water ranging from moderate to excessive quantities. The flat topography characterizing most of the mining districts in which the deposits occur permits of a convenient arrangement of the surface plant.

Access to the deposit is obtained by two shafts which are about 150 ft. apart. One shaft is used for ventilation, drainage and exit purposes,

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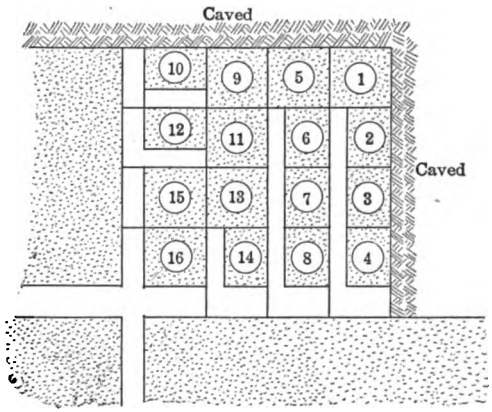
while the other is the working shaft. In a large mine the ventilating shaft is equipped with an exhaust fan, in the smaller mines natural ventilation is sufficient. Practically no dangerous gases are present and open lights are used.

The method of mining is a modification of top-slicing. The sublevel interval varies somewhat but approximates 15 ft. All of the coal with the exception of a portion 3 ft. in thickness, left in the roof of the room as a protection from the overburden and the caved cover is mined. The unit attacked is a block from 12 to 15 ft. square. A small crosscut 3.5 by 5½ ft. gives access to the room. The coal is excavated by pick and shoveled into small cars, which are trammed to inclines connecting the sublevels and the main or haulage levels. From 70 to 80 per cent. of the coal is won. The units are mined in sequence, the mining starting at



(Numbers show Sequence of Working Rooms)

FIG. 8.—METHOD OF WORKING A "SQUARE." FROM *Bergbaukunde*, HEISE-HERBST, VOL. I, p. 327.

the boundaries of the deposit or the edge of a panel. The line of retreat is stepped. After mining a room the props are pulled and the cover allowed to cave. Figs. 7 and 8 give details of the method.

The room is supported by three or four lines of props. A lid or head board is used on each prop and the sides of the room where necessary are supported by light lagging held in place by sprags. The props range in size from 5 to 7 in. in diameter. By means of short pieces of lagging the lid is given an area of from 3.5 to 4.5 sq. ft. From one-quarter to one-third of the area of the roof of the room is thus supported. The development drifts and crosscuts are closely timbered although the timbers are usually of small section. Main haulage ways and shafts are supported by masonry. The shaft sections are circular or elliptical. I saw but one timbered shaft of rectangular section. At a small mine close to Leipsic (the Dölitz) the working shaft was 75 m. in depth and of this the

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top 15 m. was brick, the next 50 m., cast-iron tubbing and the lower 10 m. brick. The shaft was 4.5 m. in internal diameter and contained two hoisting compartments and one manway. It gave access to a bed of brown coal about 40 ft. in thickness.

The general plan of development of a slice may be called a double-entry panel system. The large unit or panel is from 300 to 500 ft. square and is divided by drifts and crosscuts intersecting at right angles into smaller squares which measure from 40 to 100 ft. on a side. These squares are in turn attacked by the working drifts and the smaller units or rooms developed. The order of working off a square by rooms is shown in Fig. 8.

The inclines connecting the sublevels are served by chain lifts of a type similar to those used in the open pits. Main level haulage is either by the *kettenban* or the *seilban*. The latter is the continuous wire-rope haulage system. The cars used are of the same size and general construction as those used in the open pits.

Pumping is effected by direct-connected, electrically driven, single, two- or four-stage centrifugal pumps. Pump chambers are usually arched and supported by masonry. Significant features of the installation at one mine were the large sump space and the use of screens to prevent sand or any suspended matter from getting into the suction pipes. At one small mine visited the pumping equipment consisted of three steam pumps, one for continuous service of somewhat over 500 gal. per minute and two in reserve of a combined capacity of 1,500 gal. per minute. The head against which these pumps operated was 240 ft. At another and somewhat larger mine the pumping capacity in multi-stage centrifugals aggregated 3,200 gal. per minute. I was informed of a mine which was equipped with a pumping equipment of over 6,000 gal. per minute capacity. These figures will give some conception of the pumping problem. The heads against which the water must be lifted are in most cases not excessive, ranging from 100 to 250 ft.

Preparation of Coal for Market

Brown coal is graded into large lump, fist, nut and fine sizes. The first three sizes are sold without further preparation. The fine material is mixed with water and put through a common brick press. It is molded and cut into bricks about the size of the ordinary building brick. These are air dried in open sheds and sold as "nass brick." They contain about 20 per cent. moisture. The greater part of the brown-coal output is briquetted. The coal is first passed through crushing rolls which reduce it to ¼-in. size. It is then elevated and conveyed to driers, which are constructed very much like McDougall roasting furnaces. Each of the 32 hearths is covered by a steam coil. Rabble arms distribute the coal

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and transfer it from hearth to hearth. The coal enters with about 50 per cent. moisture and is discharged with about 12 per cent. It is then conveyed to bins from which it is fed to the briquetting presses. In all the plants visited there was but one type of press in use. This was driven

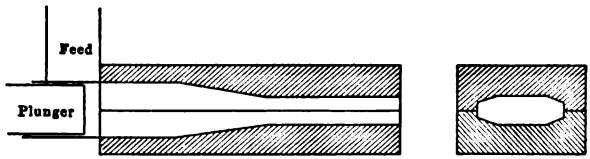

FIG. 9.—LONGITUDINAL SECTION OF BRIQUET MOLD.

by steam. The die or mold of the press is formed of two steel plates which form symmetrical halves and are firmly held in the machine. The opening between the plates narrows down from a rectangular opening of

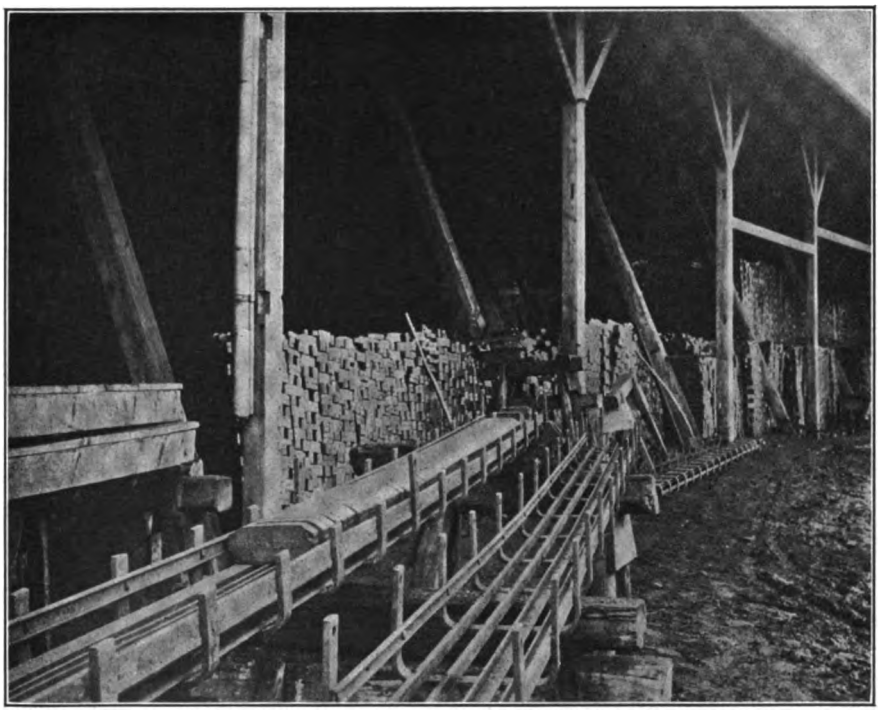

FIG. 10.—BRIQUET TROUGH AND STORAGE SHED.

the same size as the plunger to the size or section of the finished briquet, as shown by Fig. 9. A feed spout delivers the fine coal at the large end of the die, which is steam heated, and the plunger gradually forces it through the die. The briquets are delivered, apparently in a solid bar, but can be readily separated. The briquets are forced along a light

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metal trough, by the impulses of the press, to the storage sheds (Fig. 10). The press makes from 60 to 75 strokes a minute. A crank and flywheel control the length of the stroke. The capacity of a press is about 130 tons per 24 hours.

The briquet is 6 to 7 in. in length, 2½ in. wide and 1½ in. thick, elliptical in shape. The analysis of a briquet is given in *Coal Resources of the World* as follows: Water, 14.42 per cent.; ash, 7.10; fixed carbon, 33.85; volatile carbon, 44.63. The heating value of brown-coal briquets ranges from 4,500 to 5,000 calories.

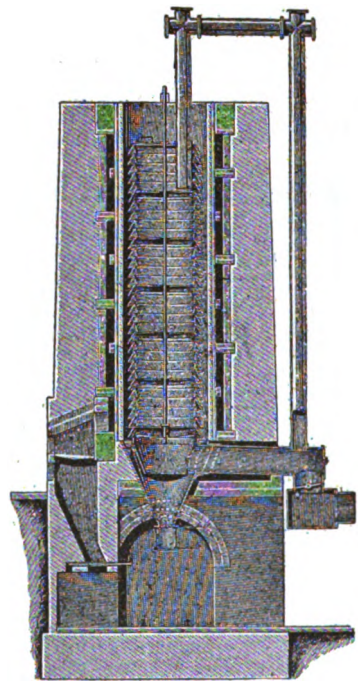

FIG. 11.—SHAFT FURNACE FOR DISTILLATION OF BROWN COAL.

In one plant visited, the Pfennershaft near Halle, part of the mine output was treated by distillation in muffle furnaces of the shaft type. The distilling chamber of the furnace is 5 ft. in diameter by 26 ft. high. The construction of the furnace is shown in Fig. 11. The brown coal is heaped over the mouth of the shaft, being prevented from entering the central part of the shaft, by a plate which covers the uppermost ring. The space between the walls of the shaft and the cast-iron rings, which are in the inner portion of the shaft, is occupied by the fine coal as it falls down. The gaseous products of the distillation pass between the iron rings and into the gas flues which reach the inner space. The combustible gases are used for heating the furnace. The fine coke, which

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is about pea size, is discharged from the hopper. The tar is used for the production of paraffin and in the chemical industries, and the coke for heating large halls, museums, etc., being extremely effective for this purpose. The coke when used for heating is burned in shallow open pans, burning without flame, very much like punk.

Surface Plants

Surface plants are compactly arranged and the buildings constructed of brick and steel. Steel headframes are used at the shafts. The smaller mines are equipped with a boiler plant, steam hoist, screening house, a brick plant for producing "nass brick," a power plant and bins. In the more elaborate plants, crushing, drying and briquetting appliances are added. Steam power is used in most all surface operations and electrical power for underground service. In one case the mine pumps were driven by steam power. Bins are constructed of steel or steel and brick.

Production, Costs, Etc.

The production in 1912 is given as 82,339,583 and in 1913 as 87,116,343 metric tons. (*Mineral Industry*, vol. xxii.) The value of the product for 1908 is given (*Berk Hütten Kalender*, 1911, p. 178) as 67,615,200 metric tons of a value of $45,230,000. This would give a value of $0.665 per metric ton. From *Braunkohle*,[1] a technical magazine devoted to the brown-coal industry, the following information was taken:

Selling price of brown coal at mine (wholesale)	$0.72 per short ton
Estimated cost of removing overburden	0.06 to 0.10 per cu. yd.
Underground mining cost, estimated	0.45 per short ton
Cost of mining coal in open pit, not including cost of stripping (coal 25 to 60 ft. thick)	0.09 to 0.10 per short ton.

The wholesale price of briquets ranges from $2.38 to $4 per ton at market centers, not at the mine. (*Bulletin* 58, *Bureau of Mines*, p. 14.)

Comment

There is considerable similarity between brown-coal mining methods and the methods used in mining soft iron ores such as occur on the Mesabi Range in Minnesota. The development of mining practice in two widely separated mining fields where somewhat similar physical conditions pertain is worthy of comment. The methods used upon the Mesabi are steam-shovel stripping and open-pit mining in which the steam shovel is almost exclusively used as a loading device, the milling system and top-slicing. The methods used in brown-coal mining are

[1] Serial article, "Ueber die Bemessung des Verhältnisses zwishen Kohlen- und Deckgebirgsmächtigkeit für Tagebaubetrieb im Braunkohlenbergbau," *Braunkohle*, July 20, 1909. The figures apply to the district about Halle.

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stripping with the continuous-bucket excavator and open-pit mining, in which the loading is effected by the milling system, and top slicing. On the Mesabi a large proportion of boulders in the glacial drift and more or less rock makes it almost impossible to avoid the use of the steam shovel. In brown-coal mining the overburden is sand or soft sedimentaries and the mechanical excavator is more economical than the shovel. It is worthy of note that in districts in this country where similar material must be removed in stripping the scraper bucket or drag-line excavator is being used in preference to the steam shovel. It does not possess all of the advantages that are obtainable with the continuous-bucket excavator.

The greater hardness of the iron ore as compared with the brown coal and the greater cost of labor in the United States explain the use of the steam shovel for ore mining. The soft nature of the brown coal on the other hand would indicate that a more extended use of mechanical appliances would be advantageous for excavating the brown coal. In the score or more of pits visited only one mechanical cutter was observed and hand picking of the coal ruled.

Approaches to the German open-pit mines are invariably short steep inclines served by chain-haulage systems while in the Mesabi open pits the approaches are long gentle slopes, seldom exceeding a grade of 6 per cent. and more often approximating 2 or 3 per cent. The smaller outputs, the greater value of the surrounding land, the necessity for preparing the coal before it can be marketed, the greater economy in haulage and the smaller capital investment required are the probable reasons for almost general use of the steep incline in the German open pits. In the Mesabi open pits the railroad cars are taken into the pit, loaded, hauled out and made up into trains. The large outputs, the short mining season, and the flat nature of the orebodies are the principal reasons for the expensive approaches. There are cases where a skip-served incline and the double handling of the ore would be more economical than an approach of the ordinary type.

The underground methods are essentially the same in principle in both localities. The most interesting point is the comparison of the size of the top-slice unit and the time required in mining a unit. The comparison is given in the table.

Size of unit:	Mesabi	Brown Coal
Height, feet	13	12 to 13
Width, feet	14	12 to 13
Length, feet	50	12
Volume of unit, cubic feet	9,100	1,800
Tons mined per miner per shift	12	6 to 7
Estimated time required to mine a unit (1 shift of 2 miners per day), days	24	2
With 2 shifts of 2 miners per shift	12	1

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Two important factors in top slicing are the size of the unit and the time required for the maintenance of the unit as an open chamber. It is obvious that the harder the ore the larger the unit and the longer the time that it can be maintained open with nominal support. Mesabi ore is called a soft ore. It is drilled with augers and loosened with light powder charges. The unit is timbered with two rows of drift sets spaced at 5-ft. intervals. The unit or chamber will remain open without serious failure of the supporting timbers for 30 days. This is sufficient to mine a unit of the size given. It is estimated that from 5 to 6 per cent. of the units begin to fail before they are completely mined out. Brown coal is decidedly softer than the Mesabi ore. Obviously a smaller unit and more rapid mining are necessary. The unit is approximately one-fifth the size of the Mesabi unit and is mined out in one-twelfth the time. Fewer and lighter timbers are used.

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The Stresses in the Mine Roof

BY R. DAWSON HALL,* BROOKLYN, N. Y.

(San Francisco Meeting, September, 1915)

THE stresses in the simplest structures are often those we find most difficult to analyze. The most complex condition in mine stresses is found in simple tunnels where the roof, the sides, and the floor are a monolith. The functions of the parts are like the parts themselves not distinct and specialized, and the problems to be solved are like those in a metal structure with riveted joints or a redundancy of bars.

This difficulty explains perhaps why the condition has not been treated. But just because it cannot be discussed in its entirety is no reason why it should be treated as an action of parts with specialized functions as a roof beam with supports and a foundation. The problem cannot be ignored on the ground that it is not of sufficient importance to warrant careful consideration, because conditions of complete monolithism, of which the tunnel is the type, are found materially unchanged in room-and-pillar and in longwall work.

This unity between roof, sides, and floor, which to the coal miner is a difficult conception, really deserves a scientific appellation, and perhaps holoid (from *holos*, whole, and *eidos*, form) will serve the purpose as well as any other.

In a simple tunnel the roof, the sides, and the floor form integral parts of one and the same structure, and the distortion of one cannot be conceived without a consequent strain in the others. Thus when the roof of the tunnel droops by reason of its weight, the upper parts of the sides are drawn in because they are integrally connected with the roof and must approach each other whenever, by the sagging of the roof, the distance between any two points in it is diminished. (See Fig. 1.)

The sides in their turn operate on the floor of the holoid structure, producing a tensile stress. The writer has always been impressed with the value of soap as a means of illustrating the action of mine stresses. With that idea in mind a cake of naphtha soap measuring 4⅜ by 2¼ by 1¾ in. was taken and a tunnel was made through it 1¾ in. long, 2½ in. wide and 1¼ in. high. (See Fig. 2.) A load was then placed at the midspan of the tunnel. Eventually, the upper bar, or "roof," broke at the

*Associate Editor, *Coal Age*.

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center line and along both "ribs" of the tunnel, the breaks being approximately vertical and proceeding, as might be expected, on the ribs from the "surface" downward and at the center line from the tunnel upward, the failure being from bending moment, not shear.

This is interesting because it shows that breakage *at the ribs* is not necessarily evidence of shear. It may be only a demonstration of a holoid structure. It is the form of failure whenever coal is blasted down and not an infrequent form of roof demolition. The test on the soap tunnel further showed that as soon as rupture takes place in the roof of the tunnel, there must inevitably come a thrust on the ribs. The tension draws them together till rupture occurs and then the two roof units, in endeavoring to revolve, crowd each other and push on the opposing walls. (See Fig. 3.)

When a holoid structure, by reason of the weakness of the floor or because of a lack of adhesion between the ribs and the floor, ceases to engage the floor in its movements, then its shape as a structural element roughly resembles the Greek letter π and for want of a better name we might term the new element a pyoid structure. (See Fig. 4.) The doctors use the word for a totally different purpose with little excuse. "Pyonoid" is the word which they should use to express the attributive "pus-like."

By reversing the soap tunnel after the roof has been caved the weakness of the pyoid structure is made clear. As soon as pressure is brought on the new roof (formerly the floor of the tunnel), the two ribs are seen to recede markedly and if the floor were intact this would result in a well-developed stress in that element of the holoid structure. (See Figs. 5 and 6, showing progressive demolition.)

Probably it is well here to express a belief in the importance of the holoid. The general notion is that all the beds shear horizontally along the lines of stratification and that it is a mistake to consider the mine or even the roof as a monolith. It is true that most of the accidents in mines are due to the lack of unity in the roof. What we call draw-slate accidents are almost wholly due to this fact. Nevertheless, it is interesting to note how firmly roof and coal are usually "burned" to one another. Even when undermined and sheared on both sides the coal often fails to fall, being supported by the vertical shearing strength of the one side still attached and by the adhesion to the roof. The writer is hardly prepared to state when the holoid structure ceases to exist, and of course the time and conditions will vary with the materials under consideration.

It is obvious that with the holoid structure, the ribs being drawn together by the movement of the roof, they must tend more or less to be split vertically and in longwall they will then fall down on the advancing undercut. In certain sub-bituminous mines the writer has noted a tendency toward what he thought was a vertical shear parallel to the

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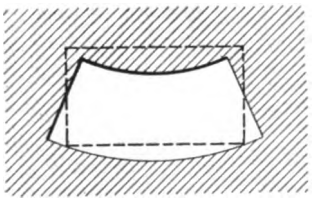

FIG.1

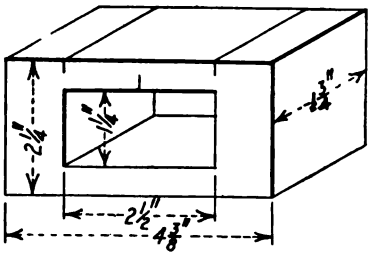

FIG.2

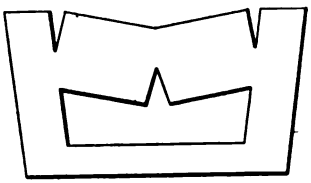

FIG.3

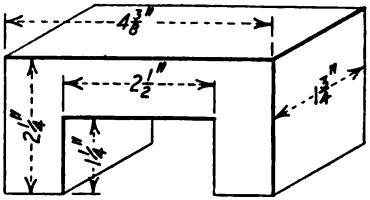

FIG.4

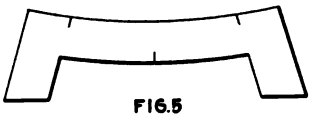

FIG.5

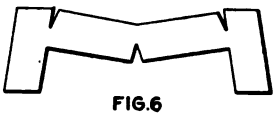

FIG.6

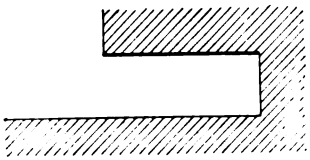

FIG.7

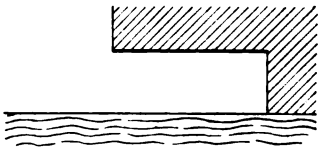

FIG.8

Fig. 1 shows in broken lines a cross-section of a tunnel and in full lines the same tunnel distorted by pressure. This is a typical holoid structure.

Fig. 2 shows a soap model to which reference is made in the article.

Fig. 3 exaggerates the upper or roof bar in the soap model and shows how the collapse of the roof forces back the sides by the arch action.

Fig. 4 shows the soap pyoid without load.

Fig. 5 shows the same when the bending of the roof bar takes place, thrusting out the ribs, or legs.

Fig. 6 shows a further action when the revolution of the segments in collapse tends to force back the upper parts of the ribs or legs.

Fig. 7 illustrates a semi-holoid structure.

Fig. 8 illustrates a semi-pyoid structure.

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headings. This developed rapidly after the work was opened, especially at great depth.

Whether this was only a vertical shear seems doubtful. It may have been due to the holoid character of the structure, the ribs receiving no relief by a horizontal shear between ribs and roof. Instead the roof pulled off the edge of the pillar as the former bent under the load. It was interesting to note that these lines of fracture did not coincide with the normal cleavage of the coal.

This rending along vertical planes eventually throws back the real rib lines far into the pillar. Where the draw slate and roof proper leave one another, we have probably a plate structure superposing one which is holoid or pyoid in character, and in the longwall the breaks back of the face which bring coal and roof down together, or which tend so to do if the latter is not duly propped, are failures of the semi-holoid or semi-pyoid and not of the plate structure above. (See Figs. 9 and 10.)

The breaking of even the holoid roof is not necessarily a sudden, unheralded event, such as one might anticipate from a cursory consideration of the problem. It is clear that the action of the moments cannot destroy the roof without revolving in a degree the elements into which the roof is broken, and any revolution inevitably binds these elements against one another so that they are less able to fall. Either a recession of the ribs or a further demolition of the revolving elements must take place or the roof will not fall. One form of demolition which frequently occurs in shallow workings is vertical shear along the cracks already made by the bending-moment stresses. But horizontal shears may make it possible for the roof masses to revolve and yet fit the space they occupied by a counter revolution of the strata past each other. Or again, the whole mass may be broken up by the rubbing of the opposing faces of the elements as they try to fall.

It is this last action which is in evidence in coal brought down by a shot when it is broken considerably in falling, and that vertical shear is not an important cause of the fall of coal is shown by the fact that there is a distinct tendency for the coal to roll away from the side ribs.

It is necessary now to consider the plate structure in which the roof is considered as a vast plate, a monolith in itself but resting without adhesion on its supports. Whenever there is a mined area the roof is depressed, and being elastic it tends to rise on the surrounding supports, resting its weight on the edges of the surrounding ribs of the excavation. This area of quasi-elevation is followed by another area of depression surrounding the central depression and the area of quasi-elevation. Thus the roof plate is bent into a series of waves around the central area of disturbance just as the surface of the water is rippled round the point where a stone has fallen and disturbed its equilibrium. (See Fig. 11.)

Of course, the elevations are only relative, not actual, and naturally,

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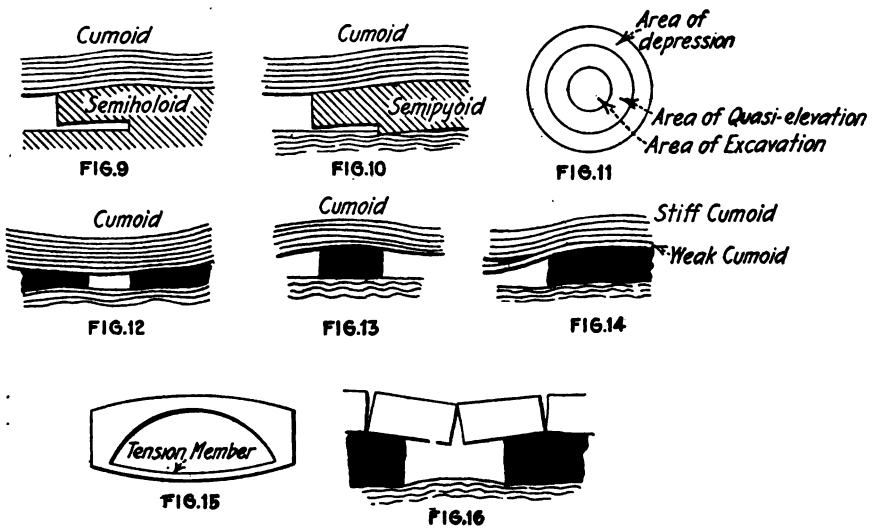

FIG.9 FIG.10 FIG.11

FIG.12 FIG.13 FIG.14

FIG.15 FIG.16

Fig. 9 shows a semi-holoid such as often occurs in longwall, supporting a cumoid which, being separated from the semi-holoid, has freedom of movement except for the restraint of frictional contact.

Fig. 10 shows a semi-pyoid likewise surcharged by an independent cumoid.

Fig. 11 is a plan of a circular excavation. The sagging of the roof into this area produces an area of quasi-elevation; that is, an area which would be elevated if the mineral itself and the floor surrounding the excavation were incompressible and indestructible. Outside this area is one of depression, the stiffness of the measures lightening the burden on the area of quasi-elevation and loading with increased heaviness the edges of the excavation and the area of depression. It exhibits the fact that the action of the roof is cumoidal, or wavelike, and not confined to the area of excavation. The depressed area is in its turn followed by another area of quasi-elevation, and this again by an area of depression. The size of these waves of course decreases as the area of excavation is left behind.

Fig. 12 shows a cumoid bending over an opening and crushing two pillars.

Fig. 13 shows a cumoid bending over a pillar, the center of which is a point of quasi-elevation. The roof tends to break over such a support.

Fig. 14 shows a weak cumoid, such as a loose draw slate which has broken away from the stiffer cumoid above and probably has insufficient moment of inertia for self support.

Fig. 15 is a Luten bridge, showing the lower tension member, or paving, which ties the abutments together and prevents their recession, thus adding to the strength of the bridge.

Fig. 16 shows a broken cumoid failing to fall owing to an arching action.

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like all undulations, as they recede from the point of disturbance they die down. But it is essential to remember that while the holoid structure is to a large extent a closed force chain, this is not nearly so true of the plate, the stresses in which are less localized and circumscribed.

This structure we may dub as cumoid (from *kuma*, a wave). The remarkable feature about such a structure is that it develops points of great stress far away from the disturbing cause, and it may break over the pillar instead of in the opening. The appearance of the Forth Bridge, Scotland, is known to almost every one. The light structural work over the midspan contrasts most forcibly with the heavy structure over the piers. It is the case with all cantilevers and continuous-arch structures, and the roof in the mine is like a continuous arch, only it is continuous not only in one, but in every direction.

It is the peculiarity of the cumoid structure that the stresses it involves may be greater farther from the point of disturbance than at some nearer point. But, like the holoid structure and the pyoid, it puts the greater burden on the pillar's edge. The center of the pillar between two large open spaces may be relieved from much of the normal pressure because of the bending of the cumoid roof over the pillar. (See Fig. 13.)

There is some evidence that in actual operations in some sections of the country the roof soon breaks by horizontal shear into two or more separate cumoid structures, of which of course one is free of external load, while the others, though below other cumoids, may or may not be loaded. If the stiffness of the upper cumoid, or cumoids, exceeds that of the lower, the loading may be relieved from the open spaces and the upper cumoids may restrain the lateral, and therefore the vertical, movement of the lower cumoids, thus adding to their resistive strength.

In an interesting paper read before the winter meeting of the West Virginia Coal Mining Institute, the late F. C. Keighley called attention to the fact that when the lower roof broke or was preparing to break in the mined spaces of the Connellsville region it frequently weakened the lower roof in the rooms and headings nearby, despite the strong support afforded by large pillars. This caused in the narrow places many falls which had to be loaded out.

It would seem, therefore, that the breaking of the lower roof or its initial stressing tears the lower roof from the upper and from the ribs, converting it into a cumoid structure which is too weak to stand the strains to which it is exposed. In cases, however, these primary failures may be due to substitution of a pyoid for a holoid structure. For it must not be forgotten that where the bottom, ribs, and roof are one and indivisible, the floor is an element of strength and prevents the roof from breaking. Just as the lower flange in a rail helps the ball of the rail to support the weight, so does the floor help sustain the roof so long as the former is unbroken.

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One of the patented features of the Luten bridge construction is its holoid structure. Luten does not use the term, but nevertheless the construction is closely analogous to the structure I have been describing. Luten builds not one bridge but two, one for vehicles to pass on and one to bind the feet of the piers together so as to form a perfect holoid. Destroy the lower bridge and the upper or traveling bridge is in a precarious condition. (See Fig. 15.)

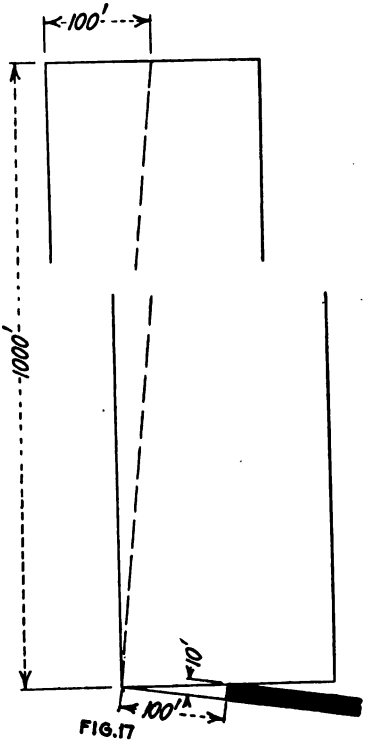

FIG. 17

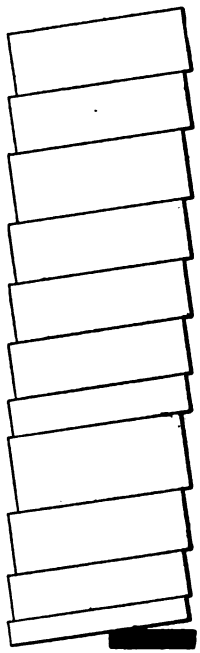

FIG. 18

Fig. 17 shows the large amount of distortion needed to permit the fall of one leaf of a broken cumoid, or conchoid, 1,000 ft. deep.

Fig. 18 shows how this distortion is secured by repeated horizontal shears at places of weakness.

The holoid structure gives the strongest of roofs; the pyoid is less strong; the cumoid is still weaker. Every horizontal shear makes the roof progressively less stable. Strange to say, vertical cracks permit the replacement in many cases of a less stable by a more stable structure. The fractures in a vertical direction, which would ordinarily be thought more detrimental than those in any other direction, prove not unimportant, it is true, yet in a way strengthening.

When a cumoid of great depth finds itself inadequate to its own support as a cumoid and begins to rend vertically from the surface down-

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ward to its supports and upward from the span centers to the surface, then the cumoid beam becomes an arch, and the cumoid plate becomes a dome and until it fails in its new structural relations, it cannot continue to extend those rents which threatened its stability. Let us say that the cumoid has become a conchoid (from *conchus*, a shell), the lines of stress forming figures resembling the shell of a bivalve.

As the elements of which the roof consists crowd each other into the opening it is impossible for the roof to rupture or fall until the various strata cease to act as a unit and begin to slide past each other. (See Fig. 16.) If we imagine an element of rock stretching from a rib to the half span, a distance, let us say, of 100 ft.; if we further suppose the excavated coal or other mineral to be 10 ft. thick, and regard the rib, floor, and falling rock element to be so adamant that none of the three will crush, heave or bend, then the inclination of the element when one corner reaches to the floor will be 10 per cent.

If the excavation is 1,000 ft. below the surface, the displacement of the block at the surface would have to be 100 ft. (See Fig. 17.) At a greater depth the displacement would be more. It is needless to say such a displacement cannot occur even where the crop, broken roof, or the sags in nearby workings make a certain amount of lateral adjustment possible. But horizontal shear along weak planes, such as beds of clay or coal or stratification planes, will permit such a number of readjustments in the element of rock itself that it will be able to come down. It will, in fact, rather *tend* to tilt than actually upset forward. (See Fig. 18.)

This, then, is how the end comes unless the rock is too strong to shear horizontally. Subsidence ceases when the required deformation has taken place. The choking up of the falls probably has but little to do with the final result, though it may have its effect in some cases. In fact, the theory as given is comforting to the conservationist, as the beds above the one extracted suffer but little, being exposed only to the horizontal shearing process, which is by no means destructive of the integrity of the measures.

However, the rupture at the half span may often result in falls of moderate size at that point, for when the monolithic roof is weakening by successive horizontal shears, we have a series of superposed cantilever cumoids entirely unequal to the task of self support. If the action which forms these cumoids does not let them all down to the floor suddenly, they will be sure to fracture successively from the bottom upward, and such seams of mineral as partake of this independent action will be destroyed.

Discussion

R. Dawson Hall, New York, N. Y.—The general practice in the past has been to view the roof as being somewhat subject only to shear. In

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fact the action of the roof under the stress of its own weight has been but little considered, not because the matter is unimportant but because it is so complicated that it has been felt that an analysis of it would not result in any conclusions definite enough to justify the inquiry. The result has been that the consideration of the subject has fallen into the hands of persons who are not experts in the study of beam and plate stress. They have been disposed naturally to view the destruction of the roof as the outcome of the simplest of stresses—vertical shear.

In the lack of a careful consideration of the conditions of roof fracture it was thought that the roof consisted of a number of horizontal layers, which were each of them weak, because no matter how strong they might be they usually lacked sufficient depth. This view of the condition of the roof would naturally lead one to expect that it would yield to bending moment, the various layers succumbing one by one to their lack of the necessary radius of gyration to meet such severe stresses. But, instead, the realization of the mutiplicity of independently strained units seems to have paralyzed the mental processes of the investigator. He realized at once that, if it were true, he had so many small units of such uncertain value to deal with that he could draw no conclusions. He therefore fell back on vertical shear in which he could regard the strata above as a unit, each element of which aided the one above to sustain the burden.

But as a matter of fact there is every reason to believe that the roof is a unit in its resistance to bending moment. Until it is broken its radius of gyration is expressible in terms not merely of the depth of individual and unit strata but in terms of the depth of the whole covering of the mineral being exploited. This possibility or rather probability has been overlooked in America, Germany, England, and France, so far as I have been able to discover, and the subject has been treated wholly as one of shear. Apart from remarks elicited in reply to my own discussion of the subject I have seen nothing on the value of the roof as a true plate until Douglas Bunting showed he recognized it, in his interesting paper on The Limits of Mining under Heavy Wash.[1] In this article Mr. Bunting treats of the consolidated material over the coal as a beam. Though he does not go into that side issue of his subject extensively, he shows a recognition of an important principle, *i.e.*, that the roof is not properly treated structurally until its resistance to bending moment has been considered.

The view that the roof was frequently a monolithic plate structure that failed first from bending moment and later as an arch and from horizontal shear was first broached in an article entitled The Strength of Mine Roofs.[2] This was further discussed at a meeting of the Coal

[1] *Trans.*, li, 177 to 199 (1915).

[2] *Mines and Minerals*, vol. xxx, pp. 474 to 476 (March, 1910).

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Mining Institute of America at Indiana, under the title Action of the Mine Roof.[3] Other discussions before the same body are Effect of Shear on Roof Action[4] and The Last Stand of the Mine Roof.[5]

CHARLES ENZIAN, Wilkes-Barre, Pa.—It would be interesting, I think, if the author would quote some authorities who have adopted the conception of mine roof as of monolithic form rather than an arch, and on what such authorities based that change.

R. DAWSON HALL.—I have never been able to find anyone who has expressed ideas similar to my own on the action of the roof. I have read many references to the subject of roof fracture. It seems the general view in France, England, and America that roof breaks from shear. The only exception I have found is the opinion of Douglas Bunting, and I did not know of his views when I formulated mine, for they were not published until long after I had come to somewhat similar conclusions myself.

There is always a natural disposition to ask for authority and proof of such theories. The conclusions are purely theoretical as far as I am concerned. Unfortunately it is almost impossible to tell what happens over the coal and under the soft surface covering which disguises such fractures, but I think if the matter is considered it will be found to be based on logical grounds and if the difficulties of giving an ocular demonstration of its truth are considered, exception will not be taken to publishing the conclusions without such confirmatory data. I have several times been asked for proof, but I can give no personal experience. Still the breaking of the roof without collapse is a significant confirmation in itself. (Mr. Keighley always declared that the roof broke on the surface half the depth of the coal in advance of the line of fracture in the mine. At Rimersburg, Pa., it is said that the surface broke over the center line of a heading, the pillars of which still stood though the room ribs were drawn right and left. Cases without number are quoted where the disturbance and breaking of the surface extends farther than the area of excavation, and the authorities allege this to be due to oblique shear or to the effect of the inclination of the measures or in some cases to a somewhat indeterminate action termed "draw.")

CHARLES ENZIAN.—I am not taking exception to the author's theory or his ideas, but I do think some reference to tests or investigations other than the few he has undertaken should be available. From the reference which he made to Mr. Bunting's paper, I thought perhaps he was familiar with some other investigations. Has not the character of the overlying strata much to do with the character of the shear?

[3] *Proceedings of the Coal Mining Institute of America*, June 28, 1911, pp. 62 to 75.
[4] *Idem*, June 26, 1912, pp. 135 to 153.
[5] *Coal Age*, vol. vi, pp. 982 to 985 (Dec. 19, 1914).

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R. DAWSON HALL.—My purpose has been to show that, generally speaking, the roof does not fail from vertical shear yet I think that there is no question but what with most shallow measures the breakage is from that stress. The smooth character of the fracture and its verticality alike attest the cause of the collapse beyond question. And it must readily be conceded that there are measures so easily sheared apart horizontally that it would be rather bold to assume that they acted under stress as a unit. Still, on the whole, I believe that the evidence shows that in most cases the monolithism of the roof is so well developed that it plays a principal part in resisting demolition.

HUGH ARCHBALD, Scranton, Pa.—It hardly matters to the miner how you theorize concerning the action of the roof as a whole, for it is the rock immediately over the coal with which he is concerned. This may be soft and come down as soon as the coal is taken away, or it may be hard and break nicely, or it may be a bending roof which never breaks. And the rocks above those immediately over the coal act in many ways.

I know that in the northern anthracite field, the roof is never treated as a monolith from the coal to the surface. Nor does it act like a monolith. For instance there is the roof over the No. 2 Dunmore seam, which is a bending roof so that when the pillars are drawn, there is no break in the mine, but bends from the pillar to the bottom. The rocks above that break and draw apart and the cracks which come are often in advance of the mining. With the roof over other seams the cracks may occur behind the mining, the rocks hanging over the edge of the pillars.

T. M. CHANCE, Wilkes-Barre, Pa.—When you are making an excavation in the coal it ought to be the same as rock; you make an excavation, and why should you not get the bending at a point 75 ft. above where the excavation is made? Why cannot you consider the rocks below an arching point as a continuous movement which is not restrained at the support? If you are going to deal with fixed continuous loads, you have one of the most obstreperous things in structural dynamics you can find. In steel work the stresses which occur in fixed continuous timbers are the subjects of argument, and have been a subject for argument for over 60 years. The only way you can consider a monolith is that the rock will have tensile stress in order to prevent the arching condition, and you will have a high tensile strength in the rock. The rock may not break; you can have an arching tendency, as in some large stopes in precious-metal mining and copper mining, where we have found an arching tendency and the roof does not arch. You must take a whole lot of material out before the arch develops. The roof is not arched, but it must arch somewhere.

R. DAWSON HALL.—The point I have endeavored to make is that, disregarding the passing phases which I have termed the holoid and pyoid

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structures, the roof resists destruction by submitting to deformation as a beam or plate. This resistance is somewhat speedily overcome as the stresses are too great for the structure to sustain, while acting in that manner. Broken as a beam, the roof finds it can support itself as an arch and doubtless as the arch fails the resistance of the only partially separated parallelopipeds comes into play again making the strains momentarily those of a beam or plate again. But with further failure the arch action again asserts itself and the load is again carried solely by the forces of compression. When the arch again fails, the tensions again come into play, each time with less power. Eventually the rock mass destroyed by tension and horizontal shears is able to sustain itself neither as a plate nor as an arch.

But it must be remembered that by an arch is meant not a physical shape but a body acted on by forces such as are made use of in the construction of an arch. Unfortunately, in discussing this subject every one instinctively thinks of the arched vaults which form in the mine as a result of breakage. But they might just as truly be arches from a structural sense if the roof had never broken. The old Mycenean arches of ancient Greece, in which the stones were laid so that each course approached the center of the span a little until the opening was bridged, were not true arches; they did not depend on compression but on tensile strength.

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The Evolution of Drilling Rigs

BY R. B. WOODWORTH,* PITTSBURGH, PA.

(New York Meeting, February, 1916)

INTRODUCTION

IN the sinking of bore holes, there are but two fundamental operations —drilling and hoisting—which determine in the main the character of drilling mechanism and structures. There are endless ramifications, however, in the execution of these fundamental operations, according to the purposes for which the bore holes are drilled, their maintenance after completion, the protection and convenience of workmen, etc.

The three main lines of use for bore holes are in mineral exploration, the sinking of water and salt wells, and the exploitation of petroleum and natural gas. This paper has to do more especially with the last. Here again there are many divergences. Wells may be drilled by either the percussive, the hydraulic, or the abrasive method. Finally, each of these methods, as applied by drillers of diverse nationalities, has followed somewhat different lines of development. The American system of cable-tool drilling has perhaps had the widest application; the hydraulic rotary is also in extensive use; but the Canadian system, the Galician system, and the Russian free-fall system all have their points of recognized merit and are preferred by operators accustomed to their use. These differences in drilling procedure, which rest ultimately on essential variations in geological conditions, are reflected in the drilling structures as well as in the drilling mechanism.

The purpose of the present paper is to record those stages in the development of the application of steel to the construction of drilling mechanism and structures with which the writer has been intimately associated, and to contribute to the history of the art of drilling other data acquired in the course of his personal investigations. It is necessarily limited by reason of space to American practice, with especial reference, therefore, to the cable-tool system and the hydraulic rotary method of drilling wells for oil and gas.

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* Engineer with Carnegie Steel Co.

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Types of Drilling Structures

With these limitations in mind we may now proceed to discuss the line of development in specific types of drilling structures. It should be borne in mind, however, that in addition to the methods of drilling and hoisting there are variations in other features, due to the arrangement of engines, the depth and size of wells, the character of strata penetrated, the preservation and maintenance of wells after completion, the means taken for the convenience and protection of workmen, and for the prevention of fire, and likewise to the character of the service and to the removal and re-use of the equipment. Drilling structures are conveniently divided into drilling rigs and drilling machines. Drilling machines are mounted on wheels and are intended for transportation, by their own motive power or otherwise, from point to point without dismantling, and their essential feature is their relatively compact character and their portability as units. Drilling rigs are either intended to remain permanently at the location in which they are first placed, or may be removed from place to place only by resolution into their component parts by dismantling and by some means of locomotion other than their own power.

Portable drilling machines have been many years in use in a number of types. The operating parts have always been made of metal, and in the conversion of their frames and other structural parts from wood into steel the line of development has been exactly parallel with that of other machinery. These machines are equipped as a rule for shallow-well drilling only, though in a number of instances quite deep wells have been put down with them. This paper has to do with drilling rigs rather than drilling machines; hence the latter will receive incidental mention only, in the course of the discussion.

Again, we distinguish between a drilling rig and an outfit. The latter comprises not only the drilling structure but also the tools, engine, boiler, ropes and other accessories. In sinking oil and gas wells by contract, it is customary for the owner to furnish the drilling rig and the casing, while the tools, engine, boilers and other parts of the outfit are furnished by the contractor.

The arrangement of the various parts of a complete drilling rig, as used in the most modern deep-well drilling for oil and gas, is shown in Fig. 1, which exhibits diagrammatically the ground plans of the three types of drilling rigs in use in the United States together with their combinations.

With particular reference to oil and gas well practice, the line of evolution in drilling structures is summarily as follows:

1. The spring pole and tripod.
2. The braced mast, single or double.
3. The four-legged braced wooden derrick intended for permanent use, built without idea of removal.

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4. The four-legged braced wooden derrick, put together with bolts to permit removal and re-erection.

5. The pipe derrick.

6. The long-panel structural steel derrick, built after the idea of what the designer thought the driller ought to use.

7. The short-panel structural steel derrick, built as nearly as possible along the lines of the braced wooden derrick.

8. The structural steel drilling rig complete, with machinery supports, wheels, house framing, metallic covering, etc.

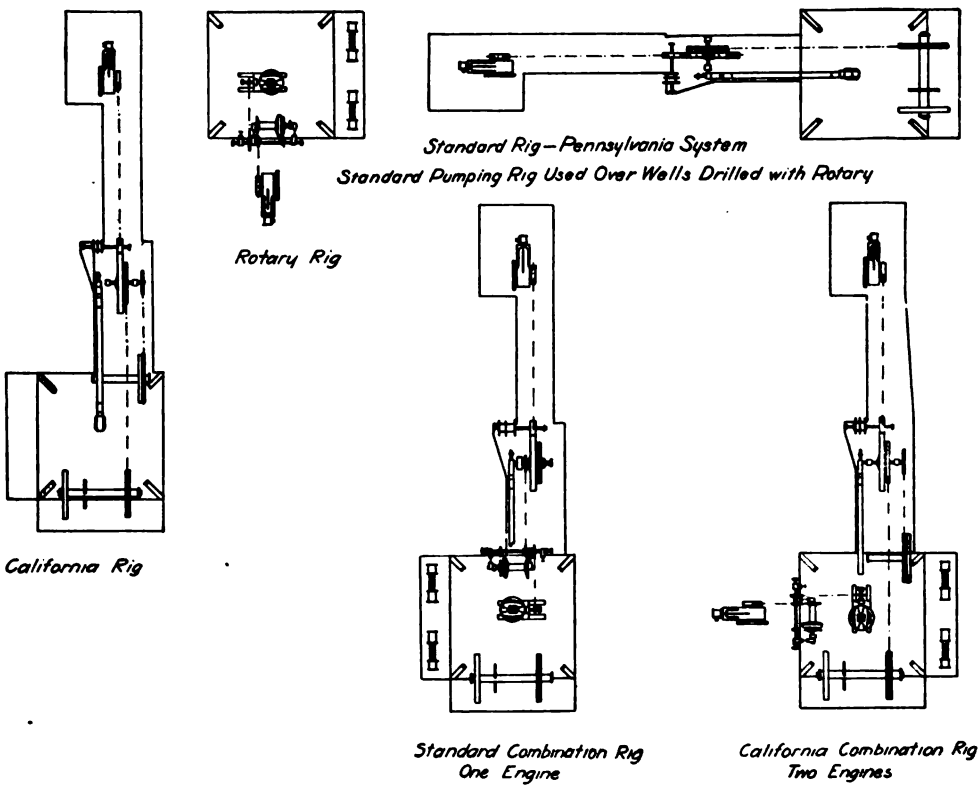

FIG. 1.—DRILLING STRUCTURES FOR OIL AND GAS WELLS. DIFFERENCES IN LOCATION OF OPERATING MECHANISM.

Our discussion will follow this division, based on the mast or derrick as the most outstanding and essential structural feature, but will also deal with incidental details of operating mechanism, so far as they are pertinent.

If in the treatment of steel derricks and drilling rigs the writer refers most largely to designs made under his supervision and executed by the company with which he is associated, it is because this company, while

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not the pioneer in the manufacture of structural steel derricks in the United States, is today the only manufacturer in a persistent and systematic way. It is also true that the complete steel drilling rig was first designed by the writer and manufactured by that company and the development of the steel drilling rig in the United States is, therefore, a record of the improvements in manufacture introduced by that company under the supervision of the writer and his superiors and assistants in office and shops, with such further collaboration and assistance as he has been able to obtain from the officials of the Oil Well Supply Co. and like companies with whom in the first instance he coöperated. The reader will pardon, therefore, any note of commercialism there may be in statements intended solely as the record of a structural development.

I. Spring Pole and Tripod

The records of well drilling in the United States begin in 1806, when the first recorded artesian well put down in the United States was drilled for salt by David and Joseph Ruffner at the Salt or Buffalo Lick near Charleston, W. Va., in the Kanawha Valley, an account of which will be found in Vol. 1A of the West Virginia Geological Survey.

1. *Spring Pole and Tripod*

Their original supply of brine came from shallow pits and the flow was more or less weakened by the presence of surface water. It occurred to them to drill wells to rock, and thus obtain a purer, stronger supply, just as in later days it occurred to George H. Bissell to abandon the shallow pits from which oil was skimmed at Titusville and tap the source with a well. A study of these two events will serve to demonstrate the analogous character of the conditions. The salt wells were the pioneers of the oil and gas industry.

The appliances of David and Joseph Ruffner were most simple. Their walking beam was a spring pole mounted on a forked stick of wood. Their bits were 2½ in. in diameter, quite primitive in construction, and attached to the end of the pole by a rope. Their tubing consisted of two long strips of wood whittled in half tubes and wrapped with twine. Their conductor was a straight well-formed hollow sycamore gum, 4 ft. in internal diameter, sunk to rock in a shallow pit. Yet the essential features of all subsequent development in the cable system of deep-well drilling were present. Jan. 15, 1808, on which day they drilled into the salt sand, is a record date in the art of drilling wells.

The first well drilled for oil was the Drake well, sunk by hand in 1859; but the next three or four wells drilled in the mad rush which followed were "kicked" down with the aid of spring poles, as were hundreds later

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in shallow territory. This method, as described by J. J. McLaurin in *Sketches in Crude Oil*, second edition, 1898, afforded a means of development to men of limited means with heavy muscles and light purses. An elastic pole of ash or hickory 12 to 20 ft. long was fastened at one end to work over a forked stick or other fulcrum. To the other end stirrups were attached or a tilting platform was secured by which two or three men produced a jerky motion that drew down the pole, its elasticity pulling it back with sufficient force, when the men slackened their hold, to raise the tools a few inches. The principle resembled that of the treadle board of a sewing machine. Fig. 3, redrawn from McLaurin, p. 76, illustrates the essential features but, like many other similar sketches, it fails to show the hoisting equipment which was indispensable even at such wells.

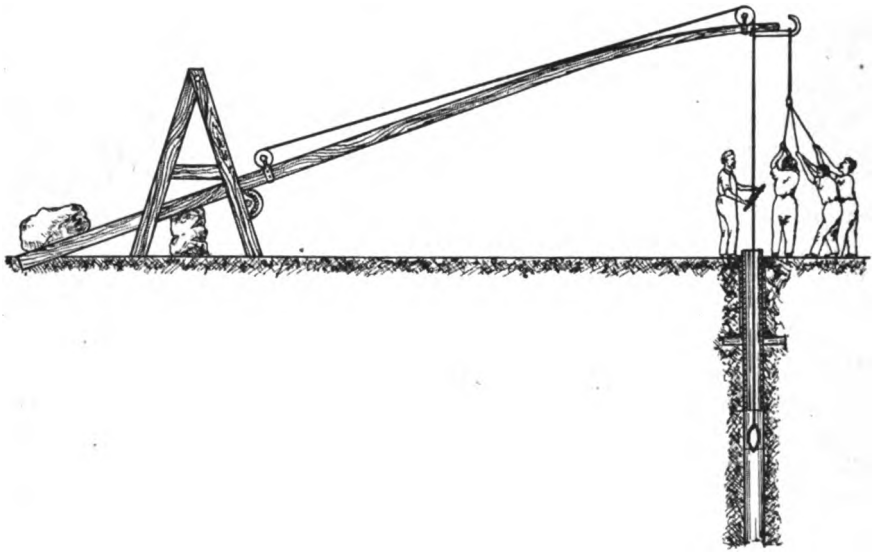

FIG. 2.—SPRING-POLE DRILLING METHOD USED BY JOBARD AT BRUSSELS, 1828. (TECKLENBURG, VOL. IV, PL. II.).

It is true that the 3-in. sand pump referred to by McLaurin could be hoisted by hand from a considerable depth; but it is obvious that, with the exception of the most shallow wells, there must have been some means of hoisting provided. W. H. MacGarvey, of Vienna, intimately connected with the establishment of the petroleum industry in Canada, and one of the founders of the great Galician-Carpathian Petroleum Corporation, who has done perhaps more than any other single man to develop the Canadian and Galician methods of drilling, has given us an exact description in the preface to J. D. Henry's *Oil Fields of the Empire*. Fig. 4, taken from this work, shows in addition to the spring pole the lower portion of the tripod with the hoisting windlass. Mr. MacGarvey says:

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"The early methods were exceedingly primitive, and at the present time many of us wonder how with such poor material such splendid pioneering results were secured. The first method of drilling in Oil Springs was by means of a spring pole. The process of putting down a well was slow and laborious, and many months were necessarily spent in drilling a well to a depth which at the present time would occupy two or three days. The first great Canadian spouter, the celebrated Shaw well, was drilled by this method."

According to publications dealing with this class of work and descriptive of operations in Canada, Ohio, New Zealand and elsewhere, the spring pole as a means of drilling oil wells disappeared about 1866. Its disappearance was due beyond question to the introduction of steam as a prime mover and the development of the portable drilling machine for the shallower wells. An excellent idea of the change and the reasons

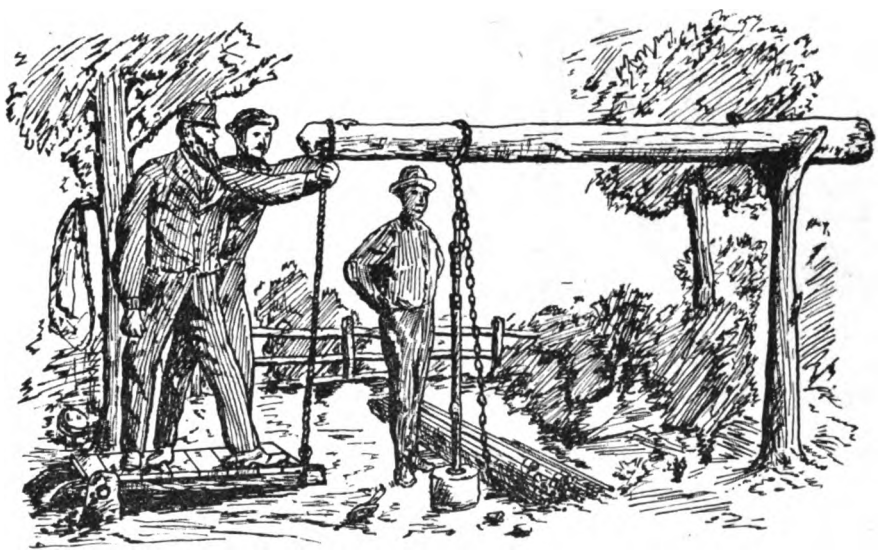

FIG. 3.—SPRING-POLE DRILLING METHOD. (MCLAURIN, p. 76.)

therefor is given by the description of the early wells drilled in New Zealand contained in J. D. Henry's book entitled *Oil Fields of New Zealand*, 1911. It was seven years after Colonel Drake drilled the first well at Titusville that the New Zealand pioneers began their search for oil with the inferior spring-pole tools of that day. The first well was sunk by Messrs. Carter, Smith, Scott, and Ross, operating as Carter & Co., at the root of the breakwater of Moturoa. On Mar. 17, 1866, the men had reached the depth of 60 ft. by shaft, but the bottom of the shaft began to feel so shaky that they decided to continue their well by boring. Spring pole and tripod were taken down and a four-legged derrick erected; and boring began at the end of 1866. The derrick bore the placard "To Oil or London," and the well was put down to a depth of 300 ft.

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The Peoples Petroleum Co. began the second or Beta well about 500 yd. from the Carter or Alpha. A 4-in. hole was sunk 516 ft. by spring pole and tripod, which bore the legend "To Oil or Edinburgh," and the well was named Victoria Well. Both wells were afterward taken over and deepened by the Taranaki Petroleum Co.

It may be noted in passing that these early wells were unsuccessful and that the first real oil produced in commercial quantities came from the No. 1 (Birthday) well of the Taranaki Petroleum Co., Ltd., completed on Sept. 24, 1906, to a depth of 2,345 ft. A steel derrick, said to be the only one then in New Zealand, was over this well at the time of Mr. Henry's visit in 1910. Early efforts to reach oil undoubtedly failed of success because of the inability of the operators to sink to the real oil-bearing strata.

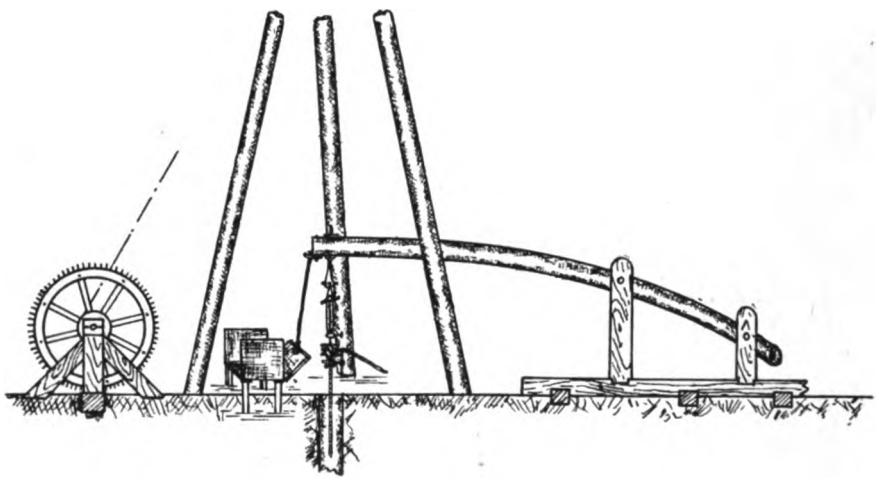

FIG. 4.—SPRING-POLE DRILLING METHOD. (W. H. MACGARVEY.)

It must not be imagined, however, that the principle of the spring pole based on its resiliency has disappeared from the art of drilling. In the fifth edition of a text-book on *Coal Mining* by Hubert W. Hughes, London, 1904, will be found on p. 17 the statement:

"In our coal districts the vibratory movement is often given to the rods by the use of the spring bar, which consists of a wooden pole having one end fixed to the ground, a fulcrum placed further on, and the rods attached to the other end. The blow is struck by depressing the beam, the rods being raised by the elasticity of it. The lengths of the parts on each side of the fulcrum are usually 1 to 3 or 5. For shallower holes the axis may be fixed, but for deeper ones it must be movable. An elaboration of this method consists in the employment of two spring poles. The first is from 60 to 70 ft. long fastened at one end and at two-thirds of its length from the fixed point it rests on an upright. To the other end are fixed two cross bars which the workmen press down on to a second spring pole, thus producing a dancing movement. Between the uprights and the cross beams is attached a hook, from which the boring tools are hung in the usual manner."

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The first edition of this book appeared in 1892 and it may be that the practice referred to was current at that date; in any event, it was not considered obsolete when the fifth edition was printed.

Moreover, the principle is now employed in connection with the boring system exploited by John R. Thom, manufacturer of boring plants at

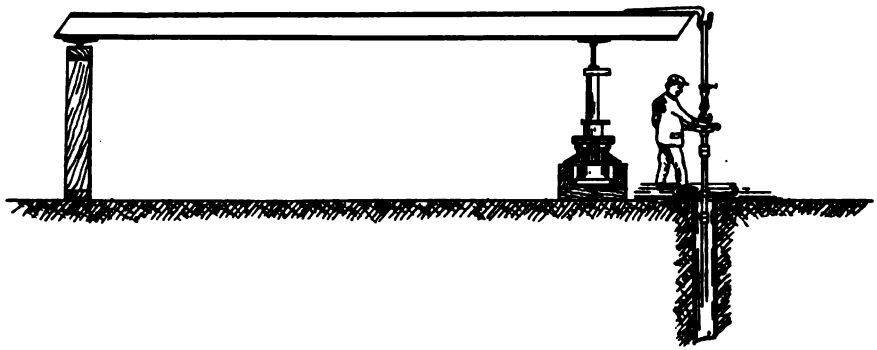

FIG. 5.—WALKING BEAM. THOM ENGLISH DRILLING SYSTEM.

Patricroft, England, as is shown by Fig. 5, re-drawn from a recent catalog. The hoisting frame used in connection with this system is a tripod with windlass mounted across two of the legs, and a sheave hung from the top. This method of hoisting is identical with that still to be found in

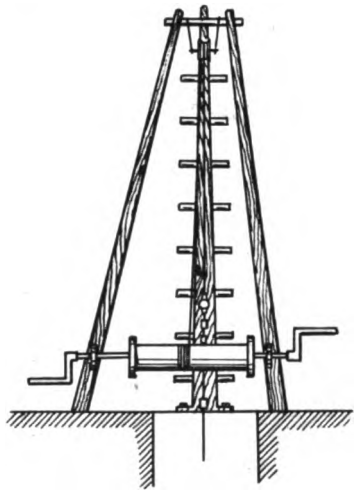

FIG. 6.—MINING SHAFT TRIPOD. (R. A. S. REDMAYNE.)

present mining practice, as shown in Fig. 6, taken from R. A. S. Redmayne's *Modern Practice in Mining*, Vol. I, London, 1908, p. 84, which is reproduced here as exhibiting the general resemblance of drilling methods whether employed in mineral exploration work or in the sinking of wells.

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The Thom system has found quite a wide application. In Vol. XXIV of the *Petroleum Review* (1911), for example, will be found a reproduction of the structure used on Plot 233 of the Maikop Oil Proprietary Co., Ltd., and on Plot 65 of the Maikop Co-operative Petroleum Co., Ltd. In connection with the latter, it is to be noted that drilling was begun by hand, and that boring contractors in that field were surprised at the rapidity with which wells could be sunk by the Thom system.

2. *Chinese Drilling Methods*

It has long been known that the art of sinking wells for salt by drilling had reached a fair state of development in China by the year 1700, and that various travelers in that country had been much impressed with

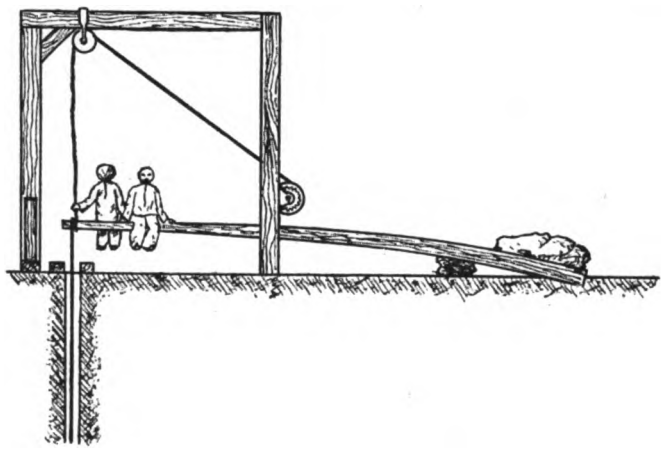

FIG. 7.—CHINESE DRILLING METHOD PER TECKLENBURG.

the ingenuity exercised in this art by the natives. Before the date named, Chinese in the district of Wu-Tung-Kiao had sunk over 10,000 wells to depths of 560 m., some even to 1,242 m. Von Richthofen narrates that the Chinese in the province of Ssu-Chan had drilled several thousand bore holes and had found salt strata at 600 m. and gas at 900 m.

Some rather fanciful descriptions have been published in the semi-technical journals. One is recalled, in which the Chinese coolies are depicted as running up over a spring board and jumping therefrom with great glee as if the business of drilling wells were recreation rather than sober industry. Fig. 7, taken from Tecklenburg's *Handbuch der Tiefbohrkunde*, Vol. IV, Plate II, is doubtless a more correct representation. It will be noted that the spring pole was counterweighted, and that the hoisting was done by a sheave suspended from a cross beam carried on a three-legged frame, the rope being wound around a windlass.

The most recent description of the petroleum, gas, and brine wells of

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Ssu-Chan, based on an original article by Louis Coldre, appeared in the December (1914) number of *Western Engineering*, published at San Fran-

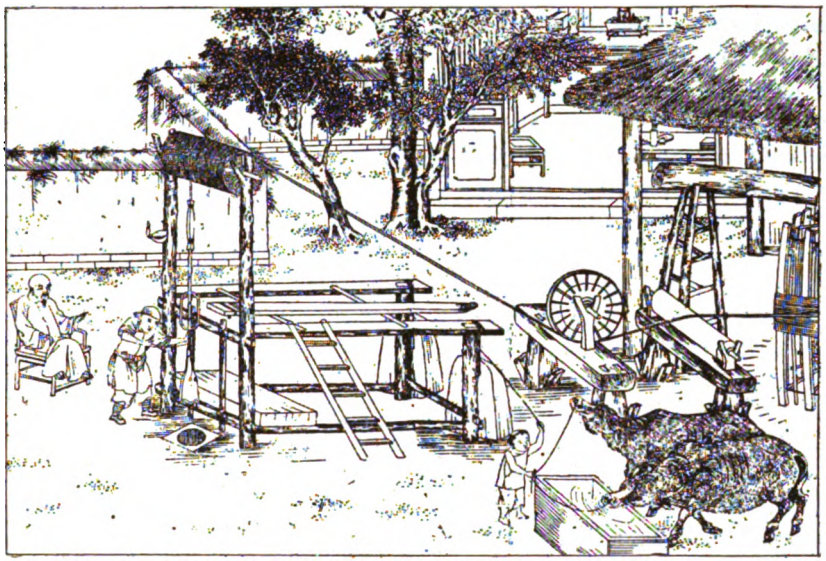

FIG. 8.—CHINESE DRILLING METHOD PER COLDRE. STARTING DRILLING.

cisco, and is well worth careful study. It indicates very great skill on the part of the Chinese. Fig. 8, taken from this paper by courtesy of the

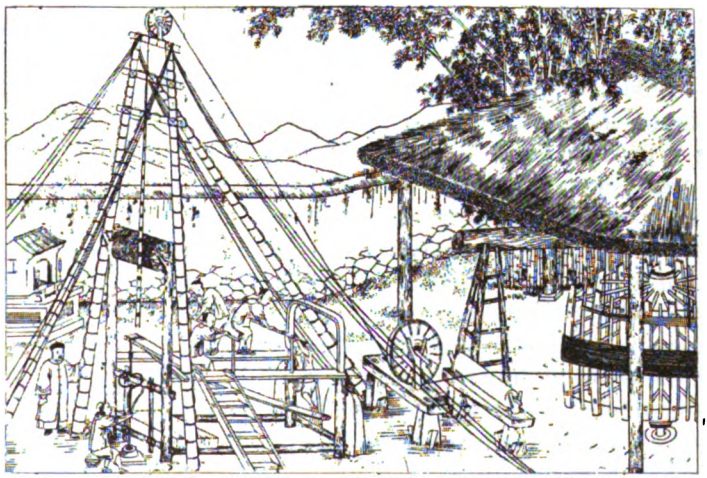

FIG. 9.—CHINESE DRILLING METHOD PER COLDRE. REGULAR DRILLING OPERATIONS.

editor, illustrates the methods of use when the wells are begun, and Fig. 9 the usual routine of regular well-drilling operations. When the well is

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started, hoisting is done by a roller carried at the top of two forked poles. When the drilling has progressed to a considerable depth, additional poles are run up and braced to carry the crown sheave placed over the header beam at the top. The line is carried down to a vertical sheave, and thence to a horizontal drum; and the string of tools is suspended from the end of a lever made of a thick plank. This is so adjusted that the lever is horizontal when the bit just touches the bottom of the hole. The four workmen then jump on the plank, depressing it and raising the bit a couple of feet. When they jump back on the side benches, the bit falls. By jumping two at a time and passing from one side to another the men are able to drill at the rate of 12 to 15 strokes per minute. The double-tripod derrick, if it may be so called, is built of bamboo and guyed with hemp ropes. The equipment, while simple, embodies the necessary mechanical features and is structurally adequate.

3. *Present Use of Tripod*

As already noted, the tripod is in modern use in the Thom system of boring. Probably its widest use is in connection with mineral exploration work, and the characteristic feature of the modern exploration outfit as used, for instance, on the iron ranges of Minnesota, is a portable drilling machine with a hoisting equipment consisting of a three-legged frame supporting a sheave at the top and carrying a windlass. Such tripods are used in heights of 18 and 23 ft., for example, in connection with the No. G drilling machine manufactured by the Ingersoll-Rand Co. They may be built of wood or of steel pipe. For deeper drilling, however, the four-legged derrick is employed.

The tripod was observed, some five or six years ago, in extensive use in the Lima fields for the ordinary routine of pulling the tubing, and the cleaning of producing wells. It was made in heights of from 30 to 40 ft. In 1907 designs were made for a 40-ft. triangular derrick to be constructed of steel for use along the same lines.

Bansen in *Das Tiefbohrwesen*, Berlin, 1912, p. 274, illustrates a triangular boring tower which is in ordinary and regular use by the General Exploration Co. in connection with its large diamond-drilling machine. This tower, built of steel pipe, five panels in height, is double tapered, the apex panel being drawn in very quickly to a point under which the single large-diameter sheave is suspended. This type of drilling tower is said to be used in sinking wells to 1,200 m. in depth.

II. Braced Mast

If the tripod is the simplest device for raising heavy weights in more or less permanent situations, the braced mast is the most convenient in

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connection with portable or semi-portable drilling equipment. The uses of the braced mast go back to the very beginning of the art. It is impracticable to trace their history in detail; and only the main line of present-day equipment will be considered here. This has two branches: first, the mast in single stick, or built up and supported on the drilling frame, the whole structure being mounted on wheels and intended for transport by its own or other power (portable drilling machine); and second, the braced mast, usually built up in parts and supported on a solid foundation, the mast being in many cases independent of drilling mechanism (semi-portable drilling machines).

1. *Portable Drilling Machines*

It was several years after the sinking of the Drake well before engines and boilers were designed especially for the superior strain of drilling and handling oil wells and thus suitable for standard-rig drilling. The Wood and Mann portable outfit, in which the small engine was mounted upon the boiler, was the popular type for the shallow territory adjacent to Oil Creek, Pa. One of these was the cause of the great fire in Cherry Run Valley above Rouseville in 1866 (*Romance of American Petroleum and Gas*, Vol. I, p. 125, 1911).

Among the earliest manufacturers in this line was the Pierce Well Excavator Co., with machines worked out about 1880 to 1890, as pictured by Tecklenburg, Vol. IV. These were arranged for transport, and also for operation, by horse-power. The mast, braced back to the machine by two braces, carried a drilling pulley at its extreme top, and a sand-line pulley part way up. Drilling was done by cable without walking beam. About 1890, the appearance is noted of that type of drilling machine usually associated with the name Star, and manufactured, according to Tecklenburg, by Gould and Austin, with the observation that the chief purpose of this very effective apparatus was to bore for water in the dry, treeless plains of Western North America. The first condition was, therefore, to replace the high timbered boring tower by an easily transportable and yet solid boring mast. These machines were suitable for depths of 450 m.

a. Side Walking Beam, Center Post.—The drilling machines made by the Star Drilling Machine Co., Akron, Ohio, embody today the same essential features of a single center mast braced to the side of the structure, and a walking beam supported at the side of the machine, the whole mounted on a wooden frame, which carries, in addition, the operating machinery, boiler and engine, and is transportable on its own wheels. These machines are made for horse transportation or are self-tractors. In the smaller machines, made without walking beam, drilling is done by spudding with a jerk line drawn over the crown pulley (see Fig. 10).

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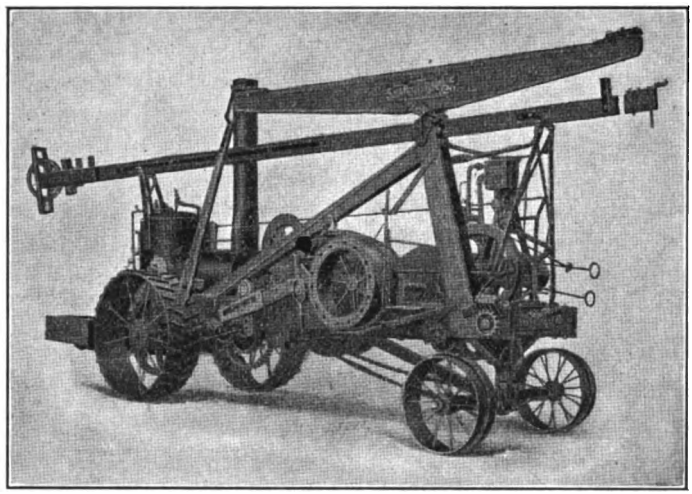

FIG. 10.—STAR DRILLING MACHINE. HEAVY SERVICE TRACTOR.

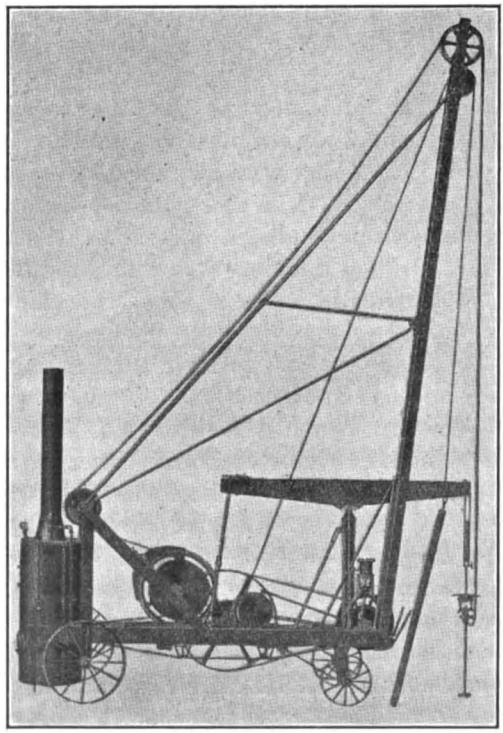

FIG. 11.—ST. LOUIS CENTER-BEAM DRILLER.

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The same structural features appear also in the Keystone driller, manufactured by the Keystone Driller Co., Beaver Falls, Pa., though in the larger machines of this type the mast is not set on the frame but is placed on a footing piece on the ground, or on the floor framing of the well.

b. Side Post, Center Walking Beam.—The St. Louis Well Machine & Tool Co., St. Louis, Mo., established about 1880, makes a drilling machine in which, as contradistinguished from the Star, the walking beam is mounted on the center, while the mast is placed on the side of the machine. The masts are usually built of wood, and the machines are made under the Robbins patent (see Fig. 11).

c. Braced Mast, Center Walking Beam.—This type of construction is an extension of the preceding one, and differs therefrom in that the mast

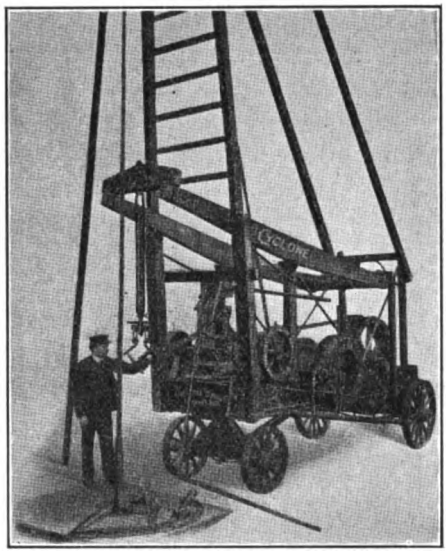

FIG. 12.—CYCLONE DOUBLE-BEAM DRILLER.

is now of the full width of the machine and the walking beam is suspended on stirrups, carried by a cross beam or suspended from the mast legs. This type of suspension is employed in the National mounted drilling machine. The Cyclone driller, manufactured by the Cyclone Drill Co., Orrville, Ohio, is in quite extensive use in all kinds of shallow-well drilling, mineral exploration, etc. The deep-well drilling machine manufactured by this company has double standard beams. These beams are supported on a samson post carried on the machine frame and reciprocate within the legs of the braced mast. The temper screw is hung from a cross bar, fastened to the outer end of the beams (see Fig. 12).

d. Braced Mast, Side Walking Beam.—The American Well Works, Aurora, Ill., was established in 1868 by M. T. and M. C. Chapman, well-

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known names in the invention of drilling machinery, particularly the rotary. Many styles of drilling machines are made by this company, most of them utilizing the jerk-line method of drilling rather than the walking beam. In the American double walking beam rock-drilling machine, however, the walking beam is made double, the mast placed within it, and reciprocating motion imparted by two pitmans, one on each side of the machine. This machine is made in two sizes and will drill 8-in. holes to 1,000 ft. The largest machines are made with steam-driven tractors (see Fig. 13).

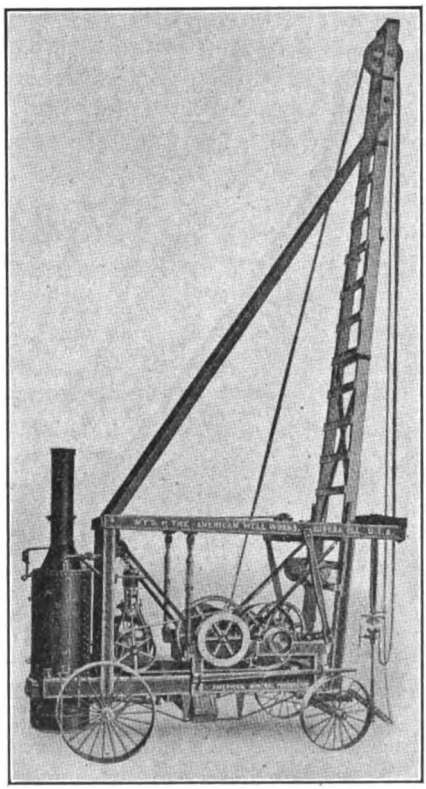

FIG. 13.—AMERICAN DOUBLE WALKING BEAM ROCK-DRILLING MACHINE.

The present Oil Well Supply Co. was organized in 1891 and is a continuation in lineal succession of the business established in 1867 by John Eaton, who is said to be the father of the oil-well supply business. The portable drilling machine manufactured by this company is the Columbia driller, formerly known as the Glenn driller, and is covered by patent No. 472,619, issued to H. S. and C. E. Glenn, Apr. 12, 1892, with later patents covering improvements. It is an all-metal machine. The mast is made of steel channels latticed together and to the full width of the

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machine at the bottom, and the walking beams, built plate-girder fashion, are reciprocated by double pitmans outside of the mast. The No. 10 Columbia weighs 18,000 lb. without boiler, and has a normal capacity of 2,500 ft. The No. 1 or Columbia Junior drilling machine is intended for water wells, test bores or blast drills, and while made with a double-braced mast, is without walking beam (see Fig. 14).

e. The Modern Self-Tractor Drilling Machine.—The essential features of this type are shown in patent No. 295,413, issued Mar. 18, 1884, to James Gillott Martin, of Bradford, Pa. The object of his invention is to provide a drilling machine capable of conversion into a traction engine, by which the machinery may be moved in smooth or moderately hilly country. Indications in the description of this patent give approximately the date of the substitution of steam for horse-power in transportation.

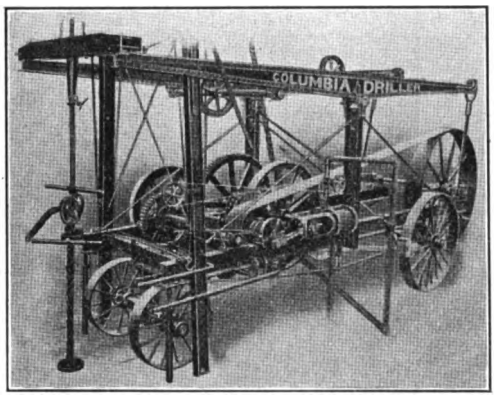

FIG. 14.—COLUMBIA DRILLER NO. 10. LOWER PART OF MAST ONLY SHOWN.

2. *Semi-Portable Drilling Machines*

A semi-portable drilling machine was patented by I. Overall, Mar. 14, 1828, utilizing the braced mast with top crown pulley as the means of hoisting. Quite a number of the modern methods in operation are to be seen in patents Nos. 46,673 and 47,729, issued in 1865 to Walter Hyde, of New York, and attention should also be called to the improved rock drill patented by J. D. Dale, of Rochester, Feb. 27, 1866, No. 52,827; also patent No. 125,355, issued Apr. 2, 1892, to Fred S. Ward and Emmert Cooper, of Theresa, N. Y. This device is particularly significant from the fact that it employed a horse treadmill as motive power. Further improvements were made in patents Nos. 257,766 and 257,767, issued to Oscar Rust, of St. Joseph, Mo., May 9, 1882.

It is believed, however, that the semi-portable drilling rig of the present owes more to George Corbett, of Bradford, Pa., than to any other single person. Patent No. 385,241, issued to him June 26, 1888, discloses

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a braced mast resting on a solid footing and carrying on its top a single sheave pulley for cleaning purposes and about half way up a larger pulley for spudding. It may be noted in passing that George Corbett is responsible for improvements in the standard drilling rig used with the four-legged braced tower, for instance as disclosed in patent No. 216,259, issued June 10, 1879. It is also to be noted that this same gentleman is responsible for the original design of the bobtail rig mentioned in discussing the special development of the steel drilling rig. The walking-beam rig ordinarily called a Corbett rig is disclosed in patent No. 396,544, issued Jan. 22, 1899, which shows in a general way a type which has come into very extensive use and consists of a wooden mast carrying two

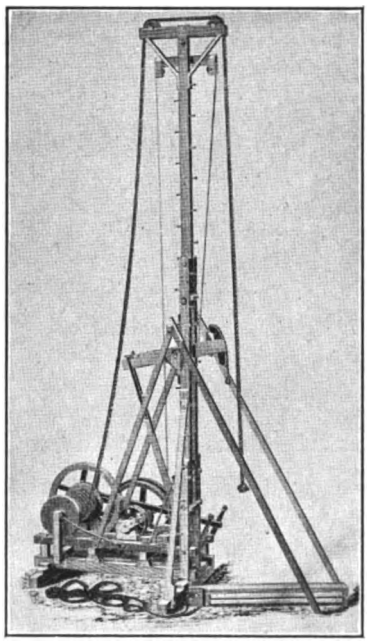

FIG. 15.—CORBETT SEMI-PORTABLE RIG.

balanced crown pulleys at its top, attachments for sand lines, and a double-braced stiff-legged pedestal. The band wheel, bull wheel, and sand reel are mounted on their own separate truck. The engine is separated from the other parts of the drilling mechanism. The band wheel is driven by a belt from the engine. The sand reel is operated by friction applied by lever; the bull wheel is also operated by friction or by a chain and clutch (see Fig. 15).

The Corbett rig was never built for wells much over 1,000 ft. in depth, but for shallow wells it came into extensive use, which continues down to the present time.

The National portable drilling rig, manufactured by the National

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Supply Co., Toledo, Ohio, and Pittsburgh, Pa., is a development of the Corbett rig and is constructed under the improvements covered by Clyde Selwyn Wright, Quaker City, Ohio, in patent No. 737,545, dated June 13, 1903. The National Supply Co. was organized in 1894, succeeding Shaw, Kendall & Co., and the Buckeye Supply Co., which had been continuously engaged in this line of manufacture since previous to 1880. The No. 2 National driller can be depended on to drill wells 2,500 ft. in depth. It is built with a mast 60 ft. high, 39 in. wide at the base, in three sections. These masts, while furnished in wood, have also been made in structural steel. The National mounted drilling machines made by the same company embody the same essential structural features (see Fig. 16).

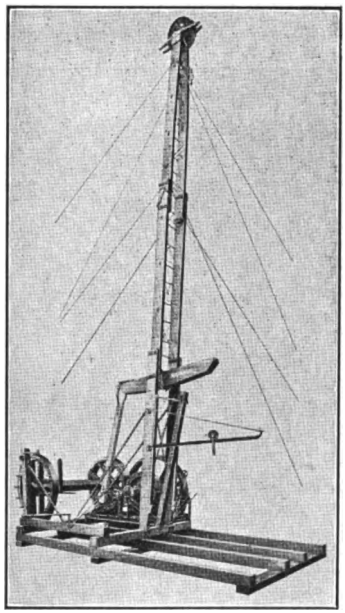

FIG. 16.—NATIONAL PORTABLE DRILLING RIG.

The same type of rig, known by the trade name of the St. Louis drilling rig, Rourke patent, is made by the St. Louis Well Machine & Tool Co., in accordance with patent No. 705,791, issued July 29, 1902, to James Rourke, of Parkersburg, W. Va.—with this difference, that in the National driller the walking beam is suspended between the legs of the mast, whereas in the St. Louis drilling rig it is mounted on its own independent samson post.

A machine of analogous character is that known as the Parkersburg chain-drilling machine, which is made with a mast 58 ft. high and is capable of drilling wells 2,000 ft. or more in depth. The operating mechanism, walking beam, and band wheel of this rig, mounted on the

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same kind of frame, may also be used with a standard four-legged bolted derrick and when so used is known as the Parkersburg bull-rope drilling machine. The chain-drilling machine made by the Parkersburg Rig & Reel Co. was introduced about 1902.

A noteworthy drilling rig of the same general character (patent No. 668,209, Feb. 19, 1901, to Charles Denio Pierce, of Jersey City, N. J.) need not detain us further here except to note that the braced mast was to be constructed of structural steel and the walking beam was to be of the same material.

3. *The Knupp Rig*

This rig, patented Mar. 14, 1905 (No. 784,571) by Jacob C. Knupp and James G. Green, of Warren, Pa., was intended to possess portability as well

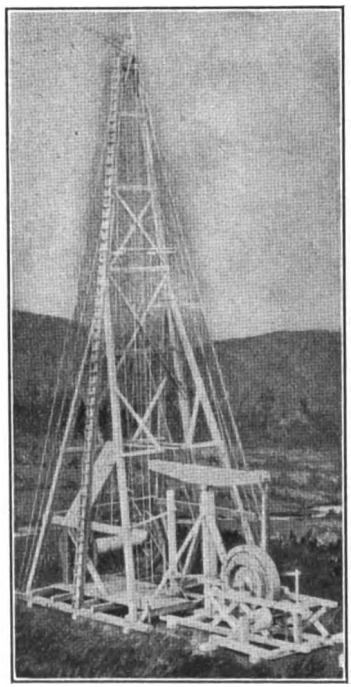

FIG. 17.—THE KNUPP SEMI-PORTABLE RIG.

as stiffness. The base of the braced mast had been widened. It was also stiffened triangularly by braces and guyed with the usual guy lines. The band wheel, sand reel, and walking beam were placed on their own independent foundation as in the Corbett rig, and this type of construction has been used in drilling wells as deep as 3,500 ft. There were unfortunate circumstances in connection with the operation of several of the rigs. Sufficient care was not taken to guy the derricks properly and

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some of them were pulled over. It is possible that a dozen would cover the number of these rigs actually placed in use about 1905. The writer is somewhat identified with the Knupp rig (see Fig. 17) for the reason that in 1903 and 1904 an endeavor was made to work up a steel structure to incorporate the lines of the Knupp construction. These propositions, however, came to naught.

III. The Four-Legged Braced Wooden Derrick

I am unable to say when the four-legged braced wooden derrick first came into use as a hoisting tower for well sinking. It is probably an evolution of the type of head frame customary in sinking mining shafts generally; and its form is so well established in the history of the art as to

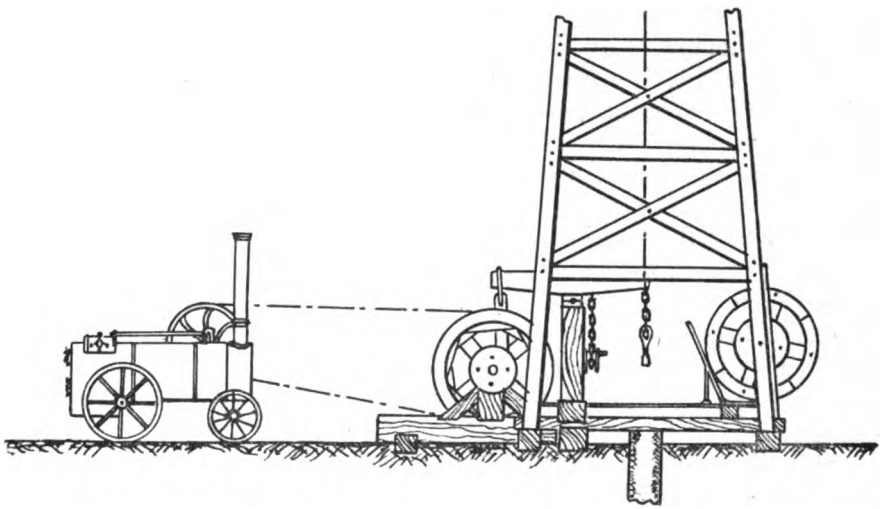

Fig. 18.—Canadian Drilling Rig, 1866. (W. H. MacGarvey.)

need no extended comment. It appears, for example, in a patent without number issued Mar. 24, 1825, to L. Disbrow, and in a more elaborate form in a patent to the same party issued Nov. 1, 1830. Unfortunately the descriptions of these two patents are not in my hands; but I note that the legs were made of square timbers, and that the girts were mortised and inserted into the legs by the use of keys, so as to provide for dismantling; also, that in 1825 the drilling line was run up over the crown pulley and attached to a treadle so as to permit drilling in treadmill fashion; and that hoisting was done by a two-handled windlass with a fly-wheel. The four-legged braced derrick made with squared timbers also appears in patent No. 47,868, issued May 23, 1865, to John Y. Smith, of Alexandria, Va.; in No. 48,006, issued May 30, 1865, to D. H. Wiswell and George W. Shaw, of Buffalo, N. Y.; and in No. 52,642, issued Feb. 13, 1866, to

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S. J. Goucher, of Philadelphia, all of which indicates that by that date the framed derrick was in such extensive use as to need no further description in patent specifications (see Fig. 18).

In the *History of Pithole*, written by "Crocus" (Charles C. Leonard) and published in 1867, it is said in description of well-sinking methods:

"First comes the derrick, a lighthouse-looking structure, some 48 ft. in height, and a temporary shed to cover it and shelter the operators and afford them sleeping quarters as it is not safe to leave the premises or machinery for even a night. If they did, they might find every movable thing had disappeared. Even the derrick and engine have been stolen in a single night and not long since an enterprising thief stole the tubing out of the bottom of the well as fast as the owner let it down to the bottom. In this manner he purloined nearly a thousand feet of tubing from a six hundred foot well."

It is true that the early oil-well derricks were constructed of poles cut from the adjoining forest and framed together in heights of 28 and 30 ft. by the use of spikes, wooden pins, and other crude devices. The ladder going up one side of the derrick was constructed by means of cleats nailed to one of the derrick legs. The essential features of girts and bracing are, however, shown on photographs taken at Pithole, Tidioute, Cherry Run, and other early oil fields in western Pennsylvania. Modifications in the four-legged derrick have been due essentially to their increase in height and to the use of sawed timber put together by carpenters instead of rough timbers cut out of wood and framed by inexperienced hands. That there is so little modification in the structure since the date of the Drake well is, beyond question, due to the fact that the essential features of the structure had already been worked out in the explorations of the salt industry.

It is, however, important to history that the four-legged braced derrick appeared on the labels pasted by Samuel Kier, of Pittsburgh, on the bottles of petroleum which he sold in 1848 and later for their wonderful medicinal virtues and that it was the sight of one of these labels and the words 400 ft. thereon which gave George H. Bissell the inspiration to undertake the organization of the Pennsylvania Rock Oil Co., under which organization Edwin L. Drake drilled the famous well at Titusville. McLaurin's statement in *Sketches in Crude Oil*, p. 81, is:

"The first derricks were of poles, twelve feet base and twenty-eight to thirty feet high. The ladder was made by putting pins through a corner of a leg of the derrick. The Samson-post was mortised in the ground. The band-wheel was hung in a frame like a grindstone. A single bull-wheel, made out of about a thousand feet of lumber, placed on the side of the derrick next to the hand-wheel, with a rope or old rubber-belt for a brake, was used. When the tools were let down the former would burn and smoke, the latter would smell like ancient codfish."

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IV. The Bolted Wooden Derrick

The advantage of wood in the construction of derricks and drilling rigs is that it is very cheap and can be made to meet any condition by building up pieces. If for any reason the rig builder makes a derrick too light and it begins to fall in, or rack to pieces, any one can take a piece of plank and nail it on in the right place to save the day. If the amount of wood used is too great for the load it has to sustain, it does not make any particular difference and does not add a great deal to the cost of the outfit, the large item of expense being in the carpenter work rather than in the material itself, though today it behooves us to pay a great deal more attention to the economical use even of wood.

This essential consideration of economy is not an absolutely new thing in this year of our Lord 1915. Beyond question, it lies back of the substitution of sawed square timber for poles and in the subsequent substitution of 2-in. plank in the legs for solid sticks of timber. We are prone to believe that the present generation invented the conservation of materials. The introduction of plank in derrick construction, say about 1865, is at once a blow to our pride and an indication of a correct though possibly unwritten appreciation of the magnitude and character of drilling stresses.

It has already been noted that L. Disbrow in 1830 intended to have his derrick put together with mortises and keys. Samuel S. Fertig, of Titusville (patent No. 130,706, Aug. 20, 1872), describes a construction and arrangement of the parts of a sectional derrick frame, whereby this frame after being built may be readily taken apart, separated into sections, or deprived of one or more sections by the removal of a few bolts without injury to the timbers, and in such manner that it may be speedily rebuilt. Fertig's derrick was made with staggered joints in two-panel lengths, with the corners, braces, and girts of plank, each part-panel piece being put together in sections and removable by bolts at the field. The use of bolts is specifically referred to in well-drilling apparatus patented by William Lowman, of Marionville, Pa., No. 388,889, Sept. 4, 1888, and is also disclosed in the patents of George Corbett. By 1892 the bolted derrick had reached such a state of development as to be actively exploited by the Oil Well Supply Co. and shown as a specific article of manufacture on p. 12 of their catalog with the comment that it is specially adapted for being moved from place to place; the various pieces being numbered and the parts which join being specially marked. In other words, there had been worked out by that date a system of standardization which would permit accurate manufacture and accurate erection. The purpose of the invention is said to be to construct, easily assemble and disassemble a boring tower, which may be sent in separate parts to timberless regions, where it can be reassembled with facility.

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The exploitation of gas has provided a field for their use much more extended than contemplated by the original invention, and the bolted derrick is, therefore, one of the most modern tools of the drilling contractor. Those made by the Parkersburg Rig & Reel Co. are perhaps in more extensive use today than any other (see Fig. 19).

The most recent development in this line is what is known in Oklahoma as the turnbuckle derrick, which is a derrick built with wooden corner posts and girts, and with iron rods fitted with turnbuckles for

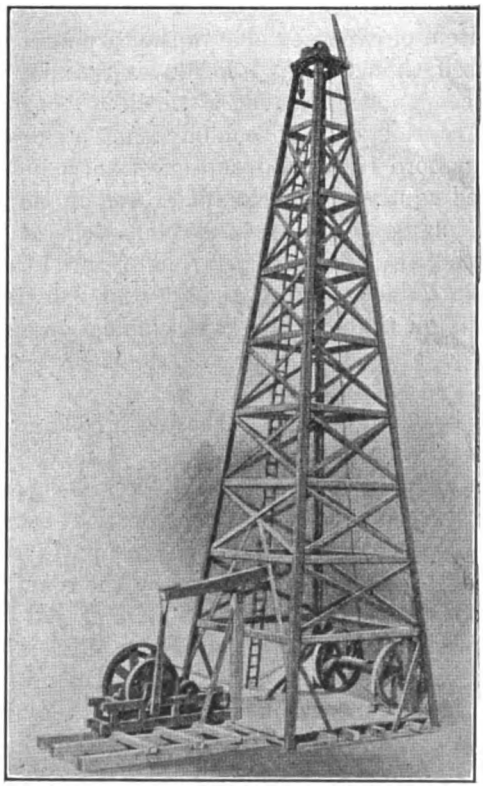

FIG. 19.—PARKERSBURG BOLTED DERRICK. TYPICAL PENNSYLVANIA CONSTRUCTION.

diagonal braces. There was said to be a growing demand for such derricks in 1913 and they are, beyond question, suitable for use in that shallow territory and for pumping purposes, in spite of the fact that the rig builders and drillers have not, as a rule, taken kindly to the use of turnbuckles. One of the most recent patents on construction of this character was No. 798,021, issued Aug. 22, 1905, to Henry Eck, of Nobleville, Ind., which seems to cover the essential features, except that the legs are made of square timbers instead of planks.

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V. Pipe Derricks

It is natural that the attention of operators should be turned early to the use of steel pipe since there are to be found in all oil and gas fields great quantities of 2, 3, and 4 in. pipe which has survived its usefulness in carrying oil, gas, or water under pressure, but is still good for structural purposes, round sections being well fitted theoretically to sustain compressive stresses. The difficulties in the construction of derricks did not

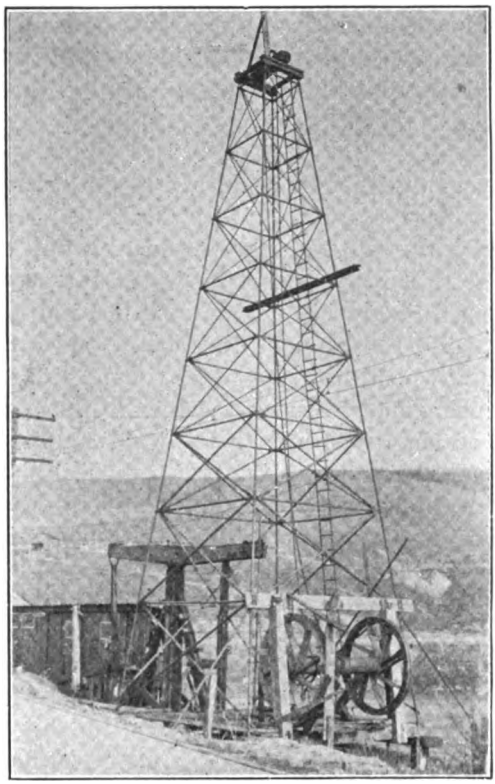

Fig. 20.—Neill Patent Pipe Derrick. MacDonald, Pa., 1909.

lie in the legs but in the other parts. Pipe is not fitted for crown blocks, or ladders, or even for the heavy girts which take the tops of the bull-wheel and calf-wheel posts. In the most modern types of pipe derricks, these parts are made of steel beams, angles, and bars. Nor is pipe suitable for bases, though it may be said, as a matter of record, that Zahniser Brothers & Sten Co., of Washington, Pa., endeavored in 1910 to make this use of it, and constructed one or two bases.

Bent pipe has also been considered for walking beams, as shown in patent No. 914,608, issued to Hezekiah Shannon, of Cyclone, an illus-

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tration of which will be given under the general heading of "Structural Steel Drilling Rigs."

Pipe is suitable for band, calf, and bull wheel shafts, but with these exceptions is, as a general rule, inadequate for the construction of more than the derrick proper. True economy requires the use of each material in the service for which it is adapted; and the use of pipe for derricks is recommended by structural fitness. The same consideration bars its use elsewhere in this line, just as it has been long since definitely abandoned in structural fabricating shops, because experience shows structural steel shapes to be more economical than pipes when both have to be made of new materials. Fig. 20 shows the Neill pipe derrick.

1. *Early Endeavors to Introduce Pipe Derricks*

Many improvements in the mechanism of well drilling were made in the early days by Jesse Button, of New York City, whose patent No. 210,007, dated Nov. 19, 1878, describes a wooden frame, constructed so as to support compactly the different parts of the machinery used in drilling wells and carrying a removable tower for lifting and lowering the drill rods. This removable tower might be made of wood or metal, but Mr. Button preferred to construct it of metal tubing, the vertical supports and cross braces being made with screw joints, so that it might be taken apart and used or not, according to the local conditions. The lower bottom panel, made of wood, was built with vertical posts, probably for the same reason as influenced the writer in the first design of the short-panel steel derrick, mentioned hereafter. The clamps at the joints are not fully described, but would probably be made of cast iron.

Patent No. 331,714, Dec. 1, 1885, was issued to Albert T. Hyde, of Oil City, Pa., for improvements in artesian-well rigs, with special reference to oil-well drilling. Mr. Hyde proposed a triangular derrick, the three legs of which were made of tubular posts, and in which the horizontal members or girts were made of pipe and the diagonal members of rods, the ends of both girts and rods being forged out flat and drilled for attachment to the legs through flanges and couplings, as shown in Fig. 21. Mr. Hyde's idea was to use one leg of the derrick to replace the samson post and so to compress the parts of a drilling rig as to place them all on one foundation about the size of an ordinary derrick floor with necessary modifications in the arrangement of engine, band wheel, bull wheel, etc. The walking beam of this rig was to be in two parts, one fastened to the derrick leg and the other to the derrick floor, and operating like a modern pumping jack.

The pipe derrick does not appear in the Oil Well Supply Co.'s catalog of 1892, which may be taken as fairly covering the state of the art at that date. It does appear, however, in the 1900 catalog, constructed

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in accordance with patent No. 346,466, granted to George Corbett, of Bradford, Pa., Aug. 3, 1886. Joints of this derrick were made in slip sockets and keyed on—no threads being used except on the turnbuckles of the stay rods, which were made of iron or steel rounds. At the joints, holes were cored in the castings to receive the ends of the diagonal rods on which dependence was necessarily placed for holding the structure together. Special castings were also provided at top and bottom for the attachment of crown block, etc.

A construction, in some respects more modern, is disclosed in patent No. 566,364, issued Aug. 25, 1896, to Amos C. Wilson, of Butler. The coupling sleeves are provided with flanged T-slots and the girts are provided with screwed-on T-heads adapted to fit the T-slots of the coupling

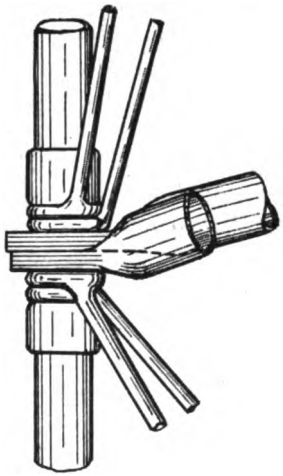

FIG. 21.—HYDE PIPE-DERRICK JOINT.

sleeves, somewhat like the way in which the cheaper grades of iron beds are constructed. Projections are formed within the circular portion of the couplings, against which the leg sections abut. Diagonal cross braces are screwed into socket braces which are connected to the coupling sleeves, through ears in their upper and lower surfaces, by bolts.

2. *More Recent Pipe Derricks*

It is impracticable at present to ascertain exactly what number of these derricks were actually constructed. It is likely, however, that some few of them came into actual use in spite of their crude design and expensive construction, just as it is likely that a few wind-mill towers were made under patent No. 490,267, issued Jan. 24, 1893, to William H. Burnham and John H. Miller, Batavia, Ill., assigners to United States Wind Engine & Pump Co., of the same place. These wind-mill towers were

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triangular and put together with 60° structural angles, clamps, and rod braces.

The use of U-bolts with or without forged clamps for fastening members of the derrick together appears also in derricks that have actually been erected and numbers of which have come into use in spite of their crude form and their relatively high cost.

At Supulpa, Okla., in October, 1909, an attempt was made by a Mr. Barnes to introduce pipe derricks, one of which, of very crude design, with screw joints and badly forged end connections, was on exhibition at Kiefer. About the same time the manufacture of pipe derricks was carried on at Washington, Pa., by Zahniser Brothers & Sten Co., who declared in their advertising literature that they had limited the use of bolts to a minimum in the construction of their make of derrick, that there were only 16 bolts used in the crown and 16 in the base, and the derrick proper could be made complete without the use of bolts; but it was believed that the use of bolts in the crown or water table made the derrick easier to take apart, if it was to be moved, as is often the case in a dry hole or gas well; and that this derrick could be taken down and rebuilt in half the time required by any other pipe or steel derrick on the market.

It was also claimed that the derrick could easily be put in perfect alignment by the use of adjustable rods (made with turnbuckles), and that it had no weak spots, since the pipes were not flattened at the ends, as in other derricks. This company also manufactured a trussed walking beam and a derrick base made of steel pipe. The design of the derrick corner coupling is similar to that patented by Amos C. Wilson but in simplified and more practical shape. Legs and girts butt against divisions in the hollow portion of the casting, and the whole structure is held together by the diagonal braces connected to bolt holes cored in the upstanding ears of the corner casting itself, as shown in Fig. 22.

Derricks made largely but not exclusively of pipe were constructed on a small scale by Simpson Brothers at Oakdale, Pa. The legs were connected by ordinary screw couplings. Girts were made of pipe flattened down 9 in. or more at each end, and with the extreme end of the flat portion bent in a quarter circle to engage the corner leg, to which it was bolted by a bolt passing through two girts, leg, and coupling. Diagonal braces were in like manner made of pipe flattened down at the ends, and connected not to the clamp but to a girt by single bolt passing through two braces and the girt (see Fig. 23). The drawing in my possession shows a 59-ft. pumping derrick only. It is obvious that this method of construction is not adequate for derricks made for drilling purposes. The ladders for these derricks are made of wood, as also are the crown blocks and bases.

Many of the derricks in use in the Evans City town-lot development, it is understood, are being made by Jacob Morris at Mars, Pa., but I cannot now say exactly what type of construction Mr. Morris follows.

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Pipe derricks are also made in the Pittsburgh district by Frank Kinney, boiler maker, at Oakdale, Pa.; and the activity of these smaller shops and shops owned not by manufacturers but by oil companies themselves, is, no doubt reflected in other places throughout the oil and gas fields, where the crude methods of manufacture are counterbalanced by the low cost of second-hand material.

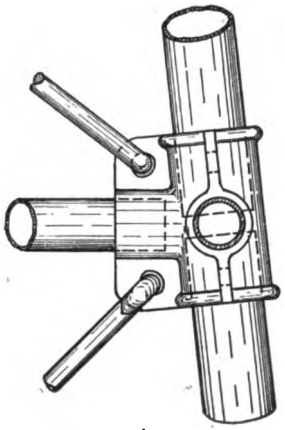

FIG. 22.—ZAHNISER AND STEN PIPE-DERRICK JOINT.

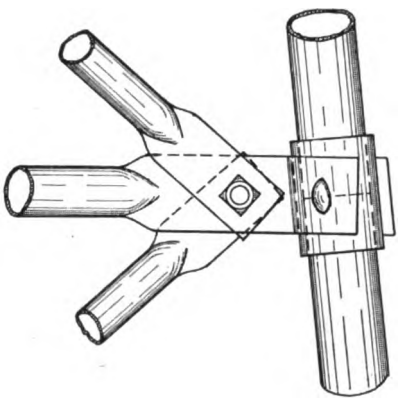

FIG. 23.—SIMPSON PIPE-DERRICK JOINT.

Among the noteworthy endeavors to utilize pipe in the construction of steel drilling rigs and derricks is a derrick made for his own operations by Godfrey L. Cabot, of Boston, Mass., the well-known manufacturer of carbon black. The method of joint construction used by Mr. Cabot is shown in Fig. 24, in which it will be noted that the splice is a malleable

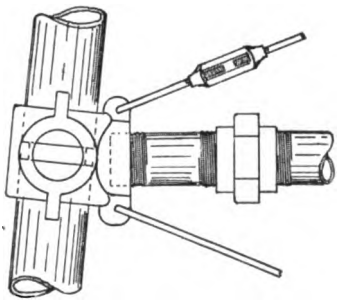

FIG. 24.—CABOT PIPE-DERRICK JOINT.

casting, with inside rings, against which the 4-in. leg sections bear, and with outside wings to which the ⅝-in. diagonal rods are attached, hook fashion. The girts are made of 2 and 3 in. pipe and are connected to shoulder nipples screwed into the joint castings by threaded ends and unions. Turnbuckles are used on the diagonal braces; and these, in

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connection with the unions, permit a great degree of adjustability in the members, which obviates the necessity for very accurate cutting to length and at the same time permits the maintenance of alignment. One of these pipe derricks erected by Mr. Cabot on the McGeary farm, some 2 miles south of Worthington, Armstrong County, Pa., has been used in drilling five holes.

3. *Neill Patent Pipe Derrick*

The development of the pipe derrick as a large commercial enterprise is due to the successful use of the forged steel clamp patented in the first instance by T. A. Neill, Field Superintendent for the South Penn Oil Co. In 1908 it occurred to Mr. Neill that a steel derrick suitable for pumping and cleaning wells could be successfully fabricated of second-hand pipe.

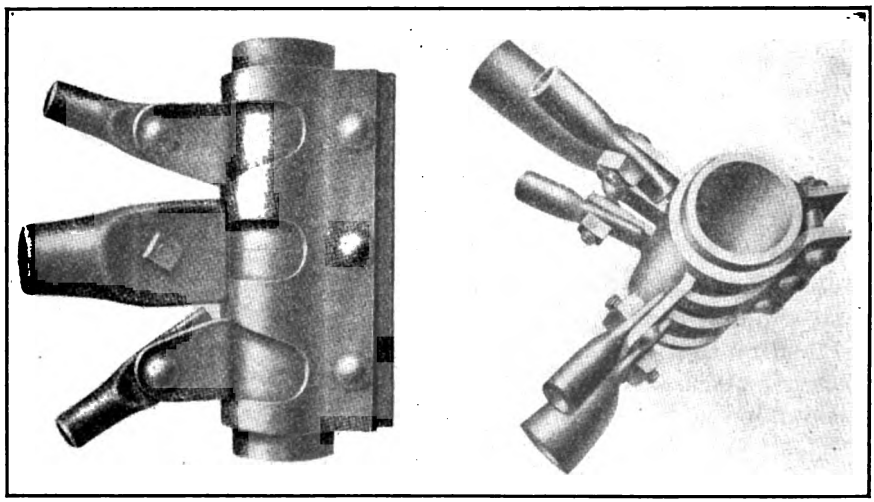

FIG. 25.—NEILL PIPE-DERRICK CLAMP.

He constructed a small wooden model indicating his idea of a clamp for the purposes of joining two pieces of pipe together without a thread and at the same time reinforcing the joint. This model was turned over to Lee C. Moore of Lee C. Moore & Co., Inc., and the first 80-ft. derrick was built in Mannington, W. Va., with clamps made of cast steel for joining the leg sections together. This first derrick demonstrated not only that the derricks would prove a success for cleaning purposes but also for heavy work as well, and Mr. Neill proceeded to cover his invention by letters patent No. 933,386, dated Sept. 7, 1909. Rights of exclusive manufacture passed to Lee C. Moore & Co., with reservation of a shop license to the South Penn Oil Co. for the construction of derricks for its own uses. Fig. 25 shows the clamping device.

During 1909 there were made and used by the South Penn Oil Co.

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and the manufacturers more than 20,000 cast-steel clamps, which were sufficient for approximately 500 derricks. Their use demonstrated, however, that cast steel was not uniform in strength and it was concluded to adopt rolled steel for the purpose. Accordingly in 1909 tools were designed to make the clamps by the drop-forging process, and since that date they have been so made. Improvements in their manufacture are covered by patents No. 1,015,821 and 1,082,207, issued to Lee C. Moore Jan. 30, 1912, and Dec. 23, 1913. The substitution of wrought steel for cast steel reduced the weight of the clamp and its cost without any decrease in the strength of the derrick at the joint, and thus largely guaranteed its success.

While the manufacturers have distributed a great many derricks made of new pipe, the more usual practice is to sell the clamps to users for the fabrication of their own derricks; and during the year 1913 the manufacturers produced sufficient clamps to construct 1,500 derricks, and that in spite of the fact that the lumber yards back of the rig builders have opposed the introduction of steel, either pipe or structural, in the construction of drilling rigs.

The extension of the Neill patent pipe derrick into rotary-drilling territory, and its use in greater heights than are required in the Appalachian oil and gas fields, made necessary the reinforcement of the pipe itself. Tests were, therefore, made under the auspices of the Bureau of Standards at Pittsburgh, which demonstrated the safe carrying capacities, under compression, of derrick leg sections built by inserting one or more tubes inside the outer section. This made it possible to increase the strength of the derrick without the use of large-sized pipe not easily obtainable and also without the multiplication of a great number of clamps, each size of which requires its own special forging equipment. It is possible to build up derricks in this manner of tubular leg sections to 30 lb. per ft. with a steel area of 8.3 sq. in.

Clamps are now made to fit all sizes of pipe from 2 to 6 in. inclusive. The heaviest derricks made of tubular sections have been of the triplex leg-section pattern (three pipes, one within the other) weighing 28 lb. per foot, 117 ft. high, 26 ft. base and 5 ft. 6 in. top. The most popular styles of this type of derrick are the 59-ft. pumper and the 80-ft. driller. These are built regularly with pipe legs, girts, and braces, tops and crown blocks of rolled-steel beams, ladders of flat steel bars and round rungs, with steel I-beam bull-wheel girts and steel channel bases, the designs for crown blocks and bases being the same as on the Woodworth derricks. By arrangement between respective manufacturers, the pipe derrick, as will be noted later, is used almost interchangeably with the structural steel derrick in the construction of complete steel drilling rigs.

The essential features of the Neill patent pipe derrick are: The design is flexible because the strength can be increased to cover any requirement

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by the use of heavier pipe for the leg section or by the duplex or triplex method of construction, and this without change in clamps. There are no screwed joints, and the derrick is assembled with clamps and bolts. The leg sections are plain, while girts and diagonal braces, which are also of pipe, are swedged down at their ends to provide a flat bearing surface against the clamp and proper width of material for punching the holes through which they are connected to the clamps themselves. U-bolts connect the girts and diagonal braces at their intersection.

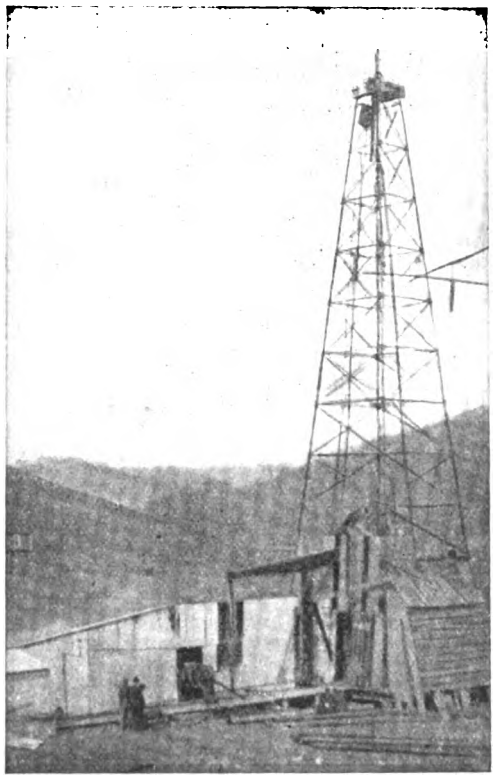

FIG. 26.—STEEL DRILLING RIG, PIPE DERRICK, 1909. SOUTH PENN OIL CO., SHINNSTON, W. VA.

It is obvious that in this type of construction the overturning of the derrick under wind or its pulling in by eccentric distribution of loads is resisted entirely by the friction of the clamp on the pipe and in consequence the bolts should be well tightened. Nine bolts are used at each joint in the 80-ft. driller and the spring action of the clamp has been found by experience to develop all the strength required. The pipe derrick is, therefore, no longer an experiment, more or less crude, but a fully developed engineering structure. Fig. 26 shows an 80-ft. derrick of this kind.

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VI. Structural Steel Derricks with Long Panels

The first derricks made of structural steel shapes were designed by men more familiar with bridge and building construction than with the practice of deep drilling. Inasmuch as the structures were supposed to remain permanently in position and to be used as structures rather than tools, convenience in erection was assumed to be a matter of secondary importance. This failure of the structural steel designer to appreciate

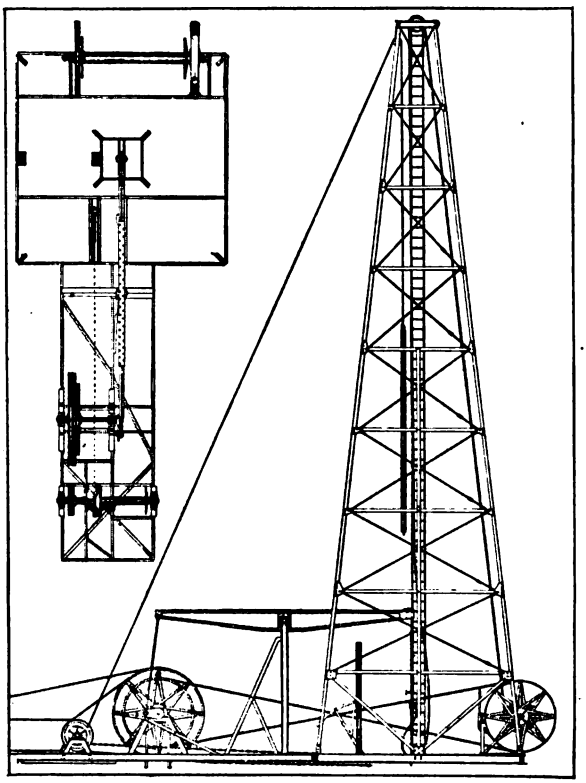

Fig. 27.—Structural Steel Drilling Rig. O. W. S. Co. Catalog, 1892.

the needs and desires of the driller has had somewhat to do with the slowness of the substitution of steel for wood.

The first appearance of the structural steel derrick known to the writer is on pp. 14 and 15 of the illustrated catalog issued by the Oil Well Supply Co., in 1892, which shows steel derricks 72 ft. high, made with steel angle legs, steel base and crown block, the diagonal members being made of rods adjustable by the use of turnbuckles. Page 14 shows the ground plan and side view of a complete steel rig with steel walking

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beam, samson post, foundation, and sectional wheels. The description of the derrick is:

"It is practically indestructible, made of light pieces easily transported and handled, which are fastened together with bolts; every part is made of steel; power is applied by a rope running through double grooves in engine pulley, band wheel and sand reel head. The first rig of this kind was sent to Australia for use in a district where wood decayed rapidly."

The transmission from engine to band wheel was by rope drive, and the sand reel was not driven, as usual, by a friction pulley in contact with the rim of the band wheel, but by a rope from the engine. The band wheel was, therefore, fitted with grooves instead of a plane surface. The impression made by the illustration is that the entire structure was primitive in form. The panels, however, were made in short length, nine in number, for the 72-ft. derrick (see Fig. 27).

The derrick reappears in the Oil Well Supply Co.'s catalog, edition of 1900, p. 13, and on p. 450 of a catalog issued in 1897 by the National Tube Works Co. in French and English, and intended for circulation abroad, with the additional information that the pieces were fitted, numbered, and painted; that the parts presented but little surface to the wind; that the wind strains had been carefully considered and that there was no danger of the wind blowing the derrick over; that the derricks were made of any height from 40 to 72 ft.; that the construction was particularly adapted for use in hot countries; and that the "first steel rig (all the parts being of metal) was sent to Australia to be used in a district where wood decayed rapidly. The outfit gave complete satisfaction and drilled a number of wells." The wheels for this type rig were exhibited at the World's Fair, Chicago, in 1893. In 1894 a second rig was built and shipped to Teheran, Persia, via Aden. Hauling from port to Teheran was via camel train.

The year 1903 marks a new endeavor to introduce the structural steel derrick. In that year the Pittsburgh plant of the American Bridge Co. built derricks in heights of 50, 60, and 70 ft., tower fashion, with three, four, and five panels respectively; that is to say, with panel heights, on an average, of 17, 15, and 14 ft. The weights of the derricks were approximately 9,000, 11,000, and 12,500 lb. The legs, braces, and girts were all angles. The ladders were made of wood. These derricks had buttresses at the base on all four sides to prevent overturning and needed for their erection gin poles or other similar scaffolding. They looked well on paper and the designer, no doubt, considered them an excellent substitute for wood. Many of them were built and some are now to be found over pumping wells in the McDonald, Pa., territory (see Fig. 28).

The same year the Chester B. Albree Iron Works, of Pittsburgh, Pa., built two or three structural steel derricks on the order of the National Supply Co. for shipment to Vera Cruz, Mexico. These derricks were 70

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ft. 8 in. high under the crown block, with a base of 21 ft. 8 in., and with buttresses of the same general character as those made by the American Bridge Co. They also had an average panel height of about 14 ft., but the ladders were of steel flats for sides and round rods for rungs (see Fig. 29).

In the same year the Carnegie Steel Co. built for the South Penn Oil Co. and the Carnegie Natural Gas Co. 12 derricks, which were used in the Pennsylvania and West Virginia oil fields. They were 80 ft. in

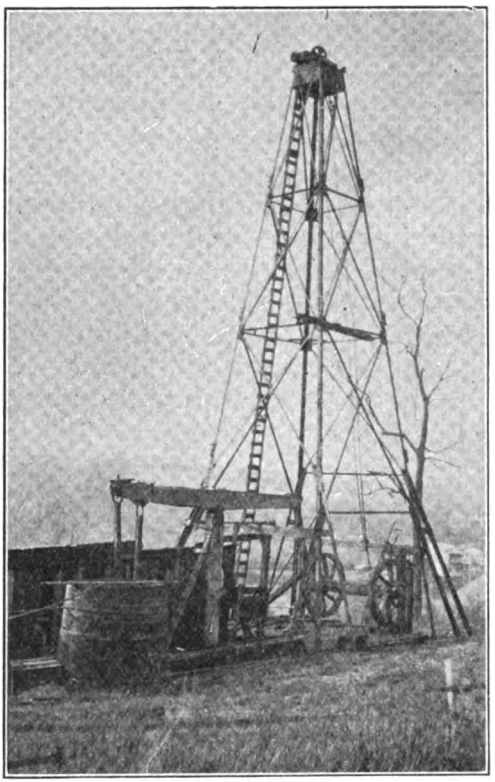

FIG. 28.—LONG-PANEL STEEL DERRICK, 1903. BUILT BY AMERICAN BRIDGE CO. ERECTED AT MCDONALD, PA.

height with a 20-ft. base and were likewise constructed along the lines of the structural steel tower, the 80-ft. derrick being built in seven panels averaging a little less than 12 ft. in height. The derrick and base weighed 24,000 lb., and provision was made at the top for a two-pulley crown block. They proved to be stiff and steady under strain; and in spite of their long panels were removed from their original location and re-erected in the drilling of more than one well each. They were probably strong enough to drill a well in this territory of a mile and a quarter deep, or

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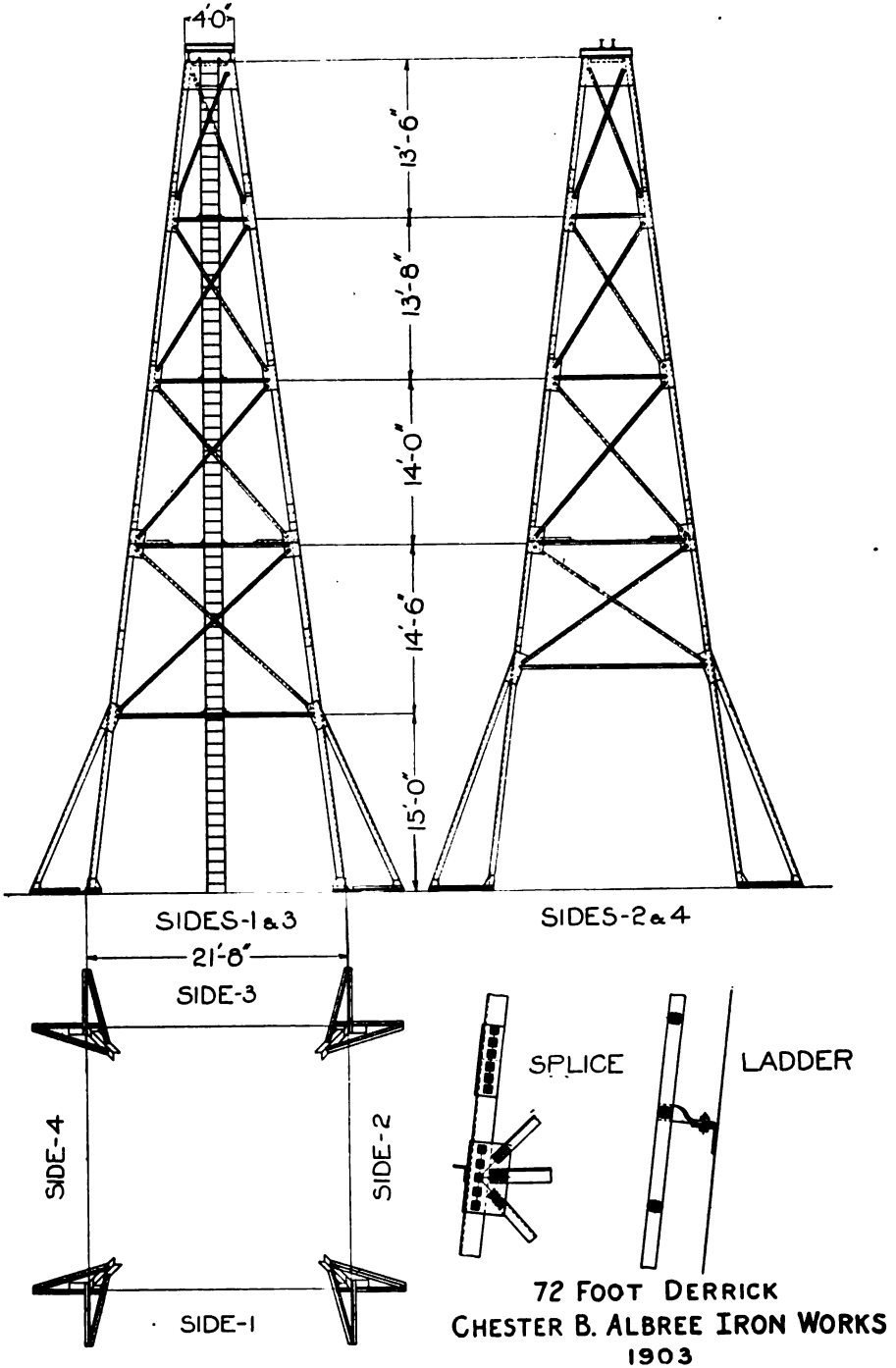

FIG. 29.—LONG-PANEL STEEL DERRICK, 1903. DESIGNED BY CHESTER B. ALBREE IRON WORKS.

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about twice as deep as any well over which they were ever erected. It may be noted in passing that the 80-ft. structural steel derrick as today constructed is built with ten panels and weighs, including crown block, ladder, and base, in the neighborhood of 15,000 lb., and will drill as deep wells as any of these 12 ever undertook (see Fig. 30).

No endeavor was made in the construction of these derricks to design a complete drilling rig. The next step in this direction was taken in 1904

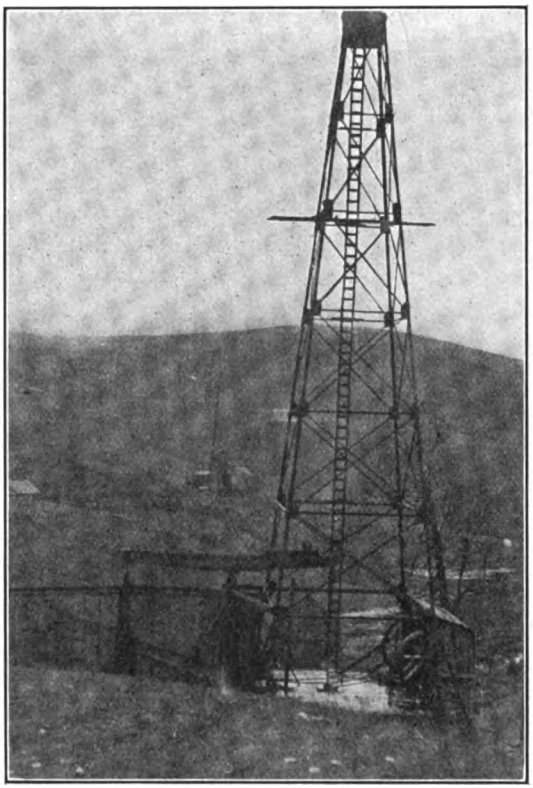

FIG. 30.—LONG-PANEL STEEL DERRICK, 1903. BUILT BY CARNEGIE STEEL CO. ERECTED BY SOUTH PENN OIL CO., SHINNSTON, W. VA.

when, in conjunction with the Oil Well Supply Co., the Carnegie Steel Co. designed a standard steel oil derrick, California type, 72 ft. high, with a 20-ft. base, the intention being to furnish, in addition to the derrick proper, the necessary machinery supports for bull wheel, calf wheel, band wheel and sand reel, and so to detail each part as to permit the use of rig irons, wheels, etc., on the steel structure without substantial divergences from parts carried in stock by oil-well supply stores and designed primarily for use on wooden structures. This derrick was likewise

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made tower fashion with six panels of an average height of 12 ft (see Figs. 31 and 42).

The first two derricks constructed after this design were shipped in January, 1908, one to Peru and the other to Argentine, after the lower panel of one of them had been erected complete in the shop for the proper adjustment of all working parts. In addition to the derrick, the order included base, machinery supports, walking beam, and house framing as far back as the sand reel, and each outfit complete weighed 34,000 lb.

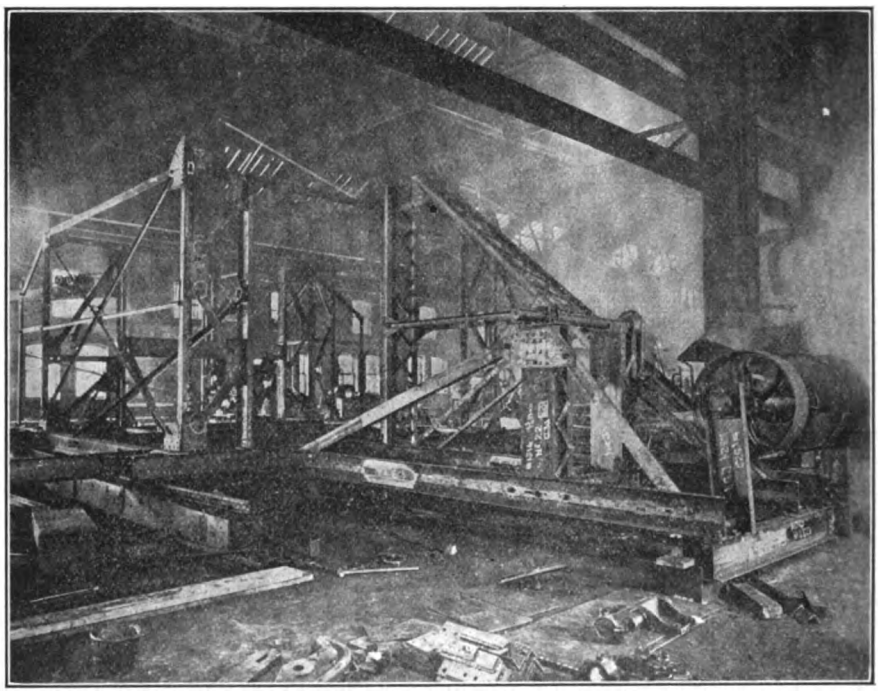

FIG. 31.—SOUTH AMERICAN STEEL DRILLING RIG, 1908. SHOP ASSEMBLEMENT OF BOTTOM PANEL.

They were shipped to New York by rail, thence by steamer to South America, and were hauled inland from 300 to 400 miles (see Fig. 32).

The performance of the Argentine rig is clearly shown in the report of the driller, written from Zapala Sept. 6, 1908, and illustrates the vicissitudes which may be encountered and successfully overcome by a well-built steel structure:

"I got the rig set up over the Acme Oil Company's drilling, a little over 1,100 feet deep, and drilled to a little less than 1,500 feet deep, when we had an English expert come out here to look over the ground and he decided on another location, and then we had to move the rig. This we did without taking down the derrick. We disconnected the sills under the sampson and jack posts from the derrick and moved each of them

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intact. The derrick we moved on 4" pipe rolling on 3 x 6" plank and pulled with block and tackle, using two yoke of oxen for motive power. The band wheel was slung on two two-wheel carts and pulled by eight oxen. The boiler was also mounted and hauled by four yoke of oxen. We moved the whole rig in four and a half days and in two days more had begun work. The derrick can be tumbled around any way and does not spring or rack in the least. It surpasses all of our expectations."

The third derrick of this design was sold to the Oil Well Supply Co. for shipment to the Burmah Oil Co., Ltd., at Rangoon.

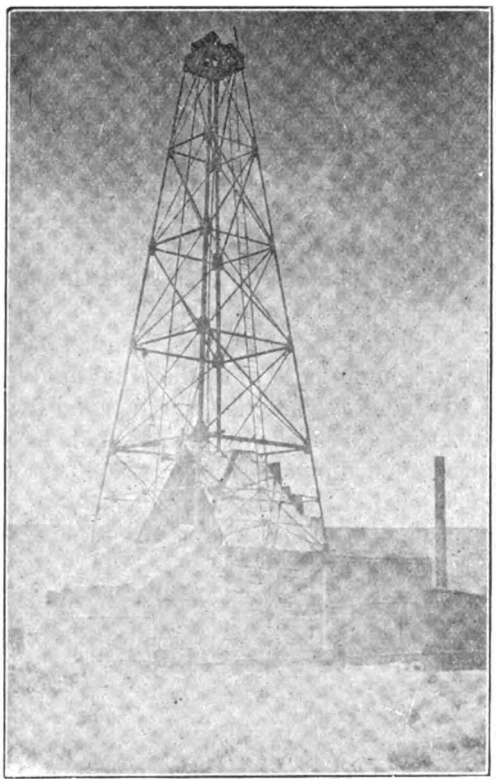

FIG. 32.—72-FT. LONG-PANEL STEEL DERRICK. BUILT BY CARNEGIE STEEL CO., 1907. ERECTED AT ZAPALA, ARGENTINE, 1908.

VII. SHORT-PANEL STRUCTURAL STEEL DERRICKS

Experience with these derricks indicated, first, that the structures were much heavier than the operating conditions in the Appalachian oil and gas fields required; and second, that the re-erection and subsequent re-use of the derricks would be very much facilitated if the panels were made short, approximately as in the wooden rig. It was further observed that, with reduced panel heights, the same strength in the legs might be obtained by the use of thinner and lighter sizes of angles (the strength of such sections

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in compression being determined by their ratios of slenderness), and the bracing made more effective.

Derricks were, therefore, designed in 1908 with short panels; and the modifications which have been made in them since have been made with a view, first, to suit the structure as exactly as may be to the wishes, and even the prejudices, of the driller, and second, to give them such strength and stiffness as was required in the light of careful observations on their use in different fields.

The designer had learned that, after all, this field of manufacture is a very conservative one; that commercial success lies in giving the driller an exact substitute for the wooden rig to which he has been accustomed; and that it is impossible to do anything more in the way of standardization than to follow only the broadest lines. Since that date the Carnegie Steel Co. has manufactured derricks to the established patterns of standard, California, and rotary types and has shipped them, on orders placed direct with the company or with dealers in oil-well supplies, to practically all the well-known fields with the exception of Europe and the Pacific Coast of the United States.

In the working out of these principles the designer has not hesitated to deviate from engineering practice as embodied in other lines of structures. He has not failed to abandon what he thought good for that which the drillers considered better. Derricks have been made as low as 40 ft. and as high as 106, with safe drilling loads from 40,000 to 300,000 lb., with capacities for the very shallowest pumping and the very heaviest drilling. Their essential feature has been only the use of structural steel wherever possible in their construction.

The short-length structural steel derrick has been manufactured with two kinds of bottom panels, in a variety of types and sizes, and with different kinds of joints. Under these three general divisions the steel derrick will be treated.

1. *Bottom Panels*

The three 72-ft. long-panel steel derricks built in 1908 deviated from the wooden derrick in that the bottom panel was made vertical instead of tapering continuously from crown block to base. The earlier designs as noted had buttresses at the bottom, because designers considered it necessary to provide extra support at the first girt, in consequence of the necessary omission of diagonal braces, the use of which is impracticable on at least the walking-beam and bull-wheel sides of a standard rig. The vertical bottom panel was an endeavor, first, to provide additional working room on the derrick floor; second, to carry out a bracing system which would make the structure self-contained; and third, to simplify the connections for machinery supports, house framing, etc., so as to reduce the expense of shop work. It was found, however, that the additional

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space was not required and in consequence the vertical panels were abandoned in 1911; records indicate that the last derrick made in this fashion was shipped in January, 1912. It should be said, however, that other designs made for derricks as early as 1908 were based on the use of a continuous taper; and, as a matter of fact, the derricks made with the vertical panel were only used in complete rigs.

2. *Types and Sizes of Derricks*

The first short-panel derricks were built for the Carnegie Natural Gas Co. in September, 1908, and were used in drilling gas wells at Waynes-

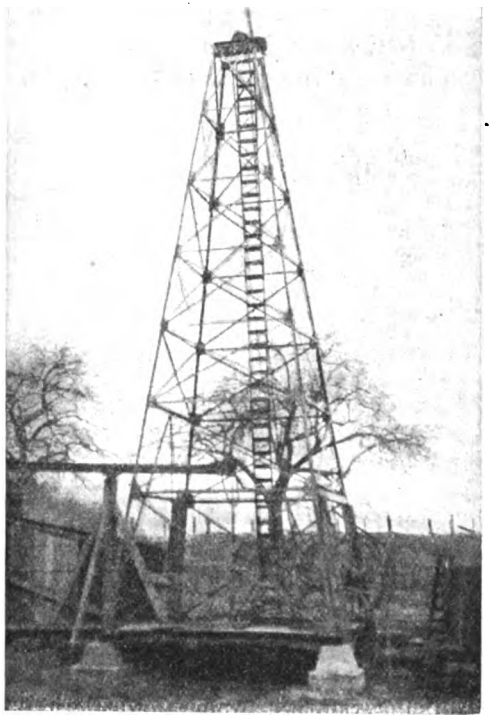

FIG. 33.—55-FT. SHORT-PANEL STEEL DERRICK, 1909. CARTER OIL CO., SISTERSVILLE, W. VA.

burg and Export, Pa., and at Burton and Hundred, W. Va.; that is, in the gas fields of Washington and Westmoreland Counties, Pa., and Wetzel County, W. Va. They were 80 ft. high and, in connection with machinery supports, made up the first steel rigs ever constructed—as will be mentioned later. It may be noted in passing that while intended for standard cable-tool drilling, the walking-beam side was constructed to

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take a calf wheel. This, however, was never used; and after the first erection the calf-wheel posts and braces were abandoned. Three more drilling rigs of the 1909 design, that is, of the pure standard type and somewhat lighter members, were built in the following year and were used by the same company in drilling gas wells in Lorain County, Ohio. These derricks have withstood safely the strains encountered in drilling wells 4,500 ft. deep and while structurally not quite so perfect as those now manufactured have been successful in every way.

In January, 1909, five 55-ft. standard derricks were built for the Carter Oil Co. and used at Sistersville, W. Va. These were short-paneled, tapered from top to bottom, and were used over pumping wells (see Fig. 33).

Otherwise sales in 1909 and 1910 were confined to 80-ft. standard derricks and 72-ft. standard and California derricks; but, as will be noted later, a new type of derrick was introduced, known as the Oklahoma, which has been manufactured in heights of 72 and 80 ft.

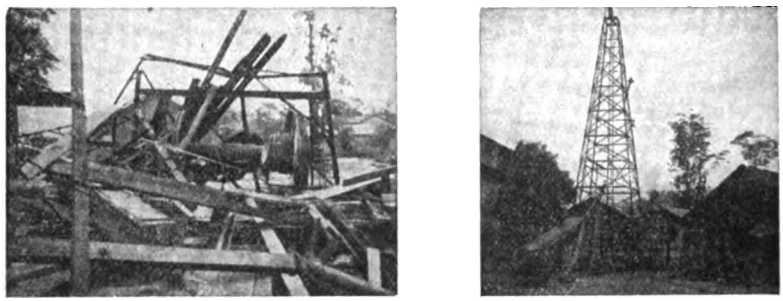

FIG. 34.—72-FT. YORKE STEEL DERRICK, 1910. HUGHES SPECIALTY WELL DRILLING CO. BLOWN DOWN AND RE-ERECTED AT STATESVILLE, ALA.

The first 80-ft. California derrick with continuous taper was made in 1911 with Yorke joints and was shipped to Cardenas, Cuba. The same year was also manufactured the first 64-ft. Yorke standard derrick for use in pumping and cleaning old wells, and also the first 64-ft. Woodworth rotary derrick. The latter was sold to the Hughes Specialty Well Drilling Co., Charleston, S. C., and has been used repeatedly in drilling water wells in Georgia, Alabama, South Carolina, and Florida. In addition to the 64-ft. Mr. Hughes also had a 72-ft. Yorke steel derrick in use in his water-well operations. This was blown down by cyclone at Statesville, Ala., straightened out and re-erected by four men in nine days at a cost of $62. Not a piece was lost; only three pieces had been actually cracked or broken and these were spliced with plates (see Fig. 34). A wooden derrick in the same storm was totally destroyed. Many of the 72-ft. California derricks manufactured in this year went to Casper, Wyo., and were used in the opening up of the new Salt Creek field.

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The rotary steel derricks were first offered for sale in July, 1911. Their introduction was due to a first-hand investigation made by the writer of drilling practice in Texas and Louisiana. The first 64-ft. rotary derrick was used for water-well drilling. The second, 72 ft. high, was made in 1912 for the Southern Well Works Co., Chattanooga, Tenn., and shipped to Gillespie & Co., San Fernando, Trinidad, for use in that field. In the same year was made the first 80-ft. California Combination rotary derrick. Four of these were made into complete rigs for the Penn-Mex Fuel Co. and were shipped to Tampico, Mexico, for use in the development of its extensive holdings on the Panuco River. In the same year also were made the first 40-ft. Yorke derricks for the Duquesne Natural Gas Co., Tulsa, Okla. These derricks, while designed primarily for pumping and cleaning, were made of sufficient strength and stiffness to pull casing.

The third field of drilling enterprise was entered in 1913 when two 40-ft. Yorke derricks were sold to the Sullivan Machinery Co. and shipped to Montivideo, Uruguay, for mineral exploration. In the same year this company purchased also a 64-ft. derrick, which was shipped to Cleveland-Cliffs Iron Co., Negaunee, Mich., for mineral exploration on the iron ranges. The derricks were furnished with a different crown block; otherwise there was no essential modification.

The first 80-ft. heavy California derrick was manufactured in 1913 and it is remarkable that the two then made were not used either for oil drilling or for water drilling or for mineral exploration, but were utilized by the Duquesne Light Co. at Neville Island, Pittsburgh, in the support of light cables and other wires across the river during the reconstruction of the Ohio Connecting Bridge of the Pennsylvania Lines West.

Four super-heavy California Combination derricks were made in the same year and used in complete rigs by the Caribbean Petroleum Co. at Maracaibo, Venezuela.

This year is also marked by the introduction of the 59-ft. rotary derricks, the first of which was shipped to the Cruse Syndicate, Port of Spain, Trinidad; the 80-ft. rotary derricks, two of which went to the same company, and others to the Southwestern Gas & Electric Co., Shreveport, La., for use in the Caddo field, and also the 106-ft. heavy Woodworth rotary derricks, the first of which was made for the Oil Well Supply Co. and shipped to the American Trading Co., Buenos Aires, Argentine. Mention should also be made of the 86-ft. heavy California derrick which was first fabricated in this same year as a part of a complete drilling rig.

The 80-ft. derrick is, as a rule, of sufficient height for ordinary cable-tool drilling as practiced in the Appalachian oil and gas fields, in Oklahoma, Wisconsin, Wyoming, etc. Recently, however, the tendency has been in the direction of longer strings of tools and casing, and especially more room is required for the heavier sheaves, hooks, etc., used in deep

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drilling. The 86-ft. heavy California derrick, therefore, constructed in 1913, was built on account of the increase in heights required by drilling improvements. It was followed in 1914 by the 86-ft. rotary derrick, the first of which went to F. E. Spence, Semmes, Ala.

The structural steel derrick, as thus developed, has had a gradual but wide distribution. It is to be found in Ohio, West Virginia, Pennsylvania, Michigan, Colorado, Louisiana, Texas, Wyoming, Oklahoma, Montana, Tennessee, Georgia, Florida, Virginia, Peru, Argentine, Uruguay, Trinidad, Venezuela, Cuba, Mexico, Ontario, Alberta, the Orange Free State, the Netherlands, and Burmah. It has been used in drilling oil wells, gas wells, water wells, salt wells, and in mineral exploration. The number of such derricks, while steadily on the increase, has fluctuated somewhat from year to year. An idea of the variety of designs required is gathered from the statistics of manufacture in 1913, in which year their use was not seriously affected by the recent depression in the oil and gas fields. Sales that year included two 40-ft. Yorke standard derricks; one 59-ft. Woodworth rotary; four 64-ft. Woodworth standard; two 72-ft. Woodworth standard; three 72-ft. Yorke standard; three 72-ft. Yorke California; 62 80-ft. Yorke standard; 15 80-ft. Woodworth standard; five 80-ft. Woodworth rotary; five 80-ft. Woodworth California; three 80-ft. heavy Woodworth California; four 80-ft. superheavy Woodworth California; one 86-ft. heavy Woodworth California, and one 106-ft. Woodworth rotary derrick. Sales that year included 30 complete rigs, of which 17 were furnished with structural steel derricks, and 13 were for use with pipe derricks. Derricks were furnished with and without bases. In addition to more or less complete structures, there were manufactured numerous miscellaneous lots of separate bases, house framing, machinery supports, bull wheels, band wheels, calf wheels, walking beams, tops, crown blocks, and girts for pipe derricks, etc.

3. *Derrick Joints*

The long-panel steel derricks were fully spliced at the joints; that is, the structural steel detail designers followed the same method of procedure as was in vogue in the detailing of structural steel towers for water tanks, etc. They put in sufficient bolts and rivets at each joint to develop the full strength of the section, and proceeded in like manner with the diagonal braces and the girts. Some of the joints were made even with both interior and exterior splices. The strength of a derrick is, of course, no greater than the strength of the weakest part; and careful consideration was, therefore, rightly given to this important feature. Experience, however, has indicated that the derrick legs seldom take anything other than direct compression and that in consequence it is safe to proceed with somewhat fewer bolts and rivets than would be considered necessary in structural steel work generally. The tendency, therefore, in the development of the structural steel derrick, as well as in the structural steel

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rig, has been to eliminate superfluous bolts and thus to reduce their number to a minimum. The splice should be of sufficient length to prevent lifting of the leg therefrom under an unequally distributed load on the crown block, and there should be sufficient bolts in the girts and braces to resist all probable wind pressures.

In the previous section, reference has been made to different types of derricks by name. These names have been adopted as a matter of commercial convenience and have particular reference to the character of the joint.

a. Woodworth Derricks.—These are covered generally by patent No. 960,474, of June 7, 1910, and have always been made with the standard structural steel splices customary in steel building construction, with the proviso above noted.

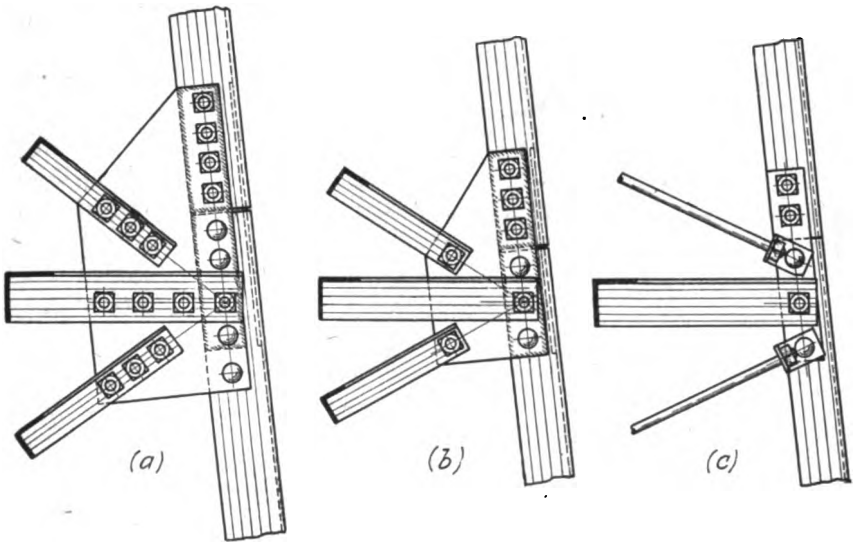

FIG. 35.—WOODWORTH DERRICK JOINTS.

The four short-panel steel derricks built for the Carnegie Natural Gas Co. in September, 1908, and subsequent derricks of the same type, were made with the joint shown in *a*, Fig. 35. The gusset plates at each intersection were riveted to the lower leg sections by eight rivets. There were inside sliding bars to form a pocket for the upper leg section. There were four bolts for each of the girts and three bolts for each of the braces, making with the bolts for the leg a total of 28 to be placed by the rig builders, or a total of 112 at each panel.

In June, 1909, the joint shown in *b*, Fig. 35, was adopted. It will be noted that the inside bar was still retained, and that wide gusset plates were still used but that the number of bolts of each intersection had been reduced to 12.

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This style of joint continued in use for about two years until Feb. 25, 1911, when the joint shown in *d*, Fig. 36, was adopted. It will be noted that now the gusset plate was riveted to the girt, that the holes in the legs were all open and that the diagonal braces were run in to take the bolts in the angle legs, the number of which has been reduced to eight. This type of joint was further simplified Aug. 29, 1912, when square-root angles were adopted for splices in the place of plates. This form of joint, shown in *e*, Fig. 36, represents the irreducible minimum. The square-root angle is riveted to the lower leg section. The girts and braces are punched with only one hole in each end, and there are but eight bolts at a joint to be put in and taken out by the rig builders, or a total of 32 at each panel as compared with the 112 which were used in 1908. This joint has

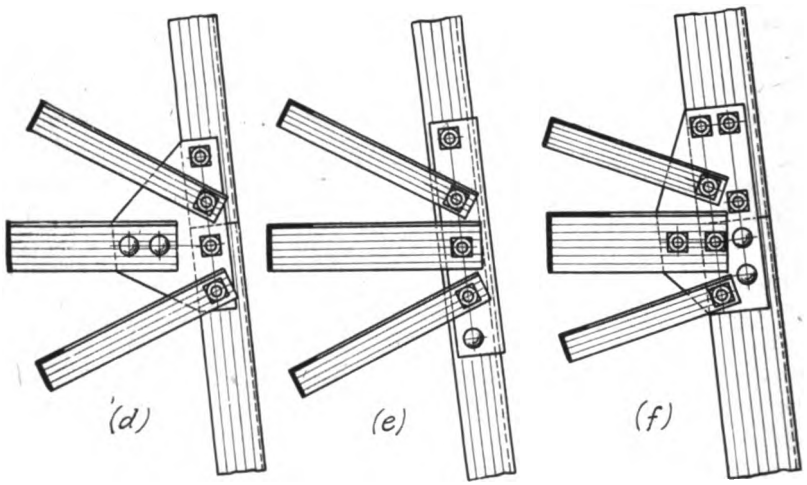

FIG. 36.—WOODWORTH DERRICK JOINTS.

given entire satisfaction, and has been amply adequate to the needs of the structure. No derrick failures have taken place in the Woodworth derricks at the joint, as indeed there have been extremely few for any cause.

The square-root angle joint last referred to is only suitable for the relatively lighter drilling derricks. The largest size square-root angle rolled by any rolling mill in the United States is 4 by 4 in., which can be used on angles with a maximum width of 5 in. The ordinary structural angles are not suitable for derrick joints on account of their inside fillets, which do not permit them to fit snugly against the derrick legs, and in consequence in all the heavy derricks with leg sections 6 in. wide or wider, it is necessary to revert to gusset plates. Fig. 36 shows in *f* the standard joint for heavy structures, consisting, at each intersection, of two plates riveted to the lower leg section by four rivets, and with all the other connections open, a total of 14 at each joint or 56 for each panel. This is the

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type of joint used on the 106-ft. steel drilling rig shown at the Panama-Pacific Exposition. It will be noted that while this derrick is figured for a safe working load of 223,000 lb. as compared with the 92,000 lb. computed as safe for the 80-ft. derrick of 1908, there are now just one-half as many bolts, whereas by the older method of computation two and a half times that many would not have appeared unreasonable. The leg sections are not faced, but the holes in the upper section are spaced on a slightly larger spacing than the holes in the gusset plates of square-root angles, and in consequence, when erected, the two sections butt smoothly and tightly together. The legs are cut by the mills with extreme accuracy, and in consequence the $\frac{1}{16}$-in. play in the bolt holes is amply sufficient to insure a solid bearing, and this solid bearing is the thing that is now relied upon, rather than the full splicing of the joints with bolts in shear, to transmit the load from the crown block to the derrick base.

b. Woodworth Oklahoma Derricks.—In 1909, the Oklahoma Iron Works secured ten derricks in heights of 55 and 68 ft., from Joseph T. Ryerson & Son in Chicago, and endeavored to introduce them in that district. So far as the writer can recall, these derricks were of rather crude construction, the girts, made of small channels, being extremely flexible, and the joints inadequate. An investigation was made of the needs of that field and in that year a very light type of derrick was placed on the market in the expectation that it would prove satisfactory in the drilling of shallow wells and in pumping and cleaning old wells.

The type of construction at the joint is shown in *c*, Fig. 35. Splices were made by means of short bars riveted to the lower leg section, above which were riveted lug angles to take the diagonal braces, which were made of round rods. These, however, were without turnbuckles. They were made with long threads and were adjusted to proper tension by the nuts at their ends. The use of these threaded rods was adopted in the first instance to reduce weight and in the second instance because drillers had not given any favorable attention to turnbuckles.

These derricks were offered for sale in heights of 55, 64, 72, and 80 ft. Seven 72-ft. derricks and six 80-ft. derricks were placed in service in 1910 and 1911 in Ohio and in Oklahoma. The use of these derricks did not commend itself to the operators, and their manufacture has, in consequence, been discontinued. They are, however, believed to be suitable for light work; but the decrease in weight is somewhat offset by increase in manufacturing cost.

c. Yorke Derricks.—In July, 1908, the writer made the acquaintance of Patrick Yorke, of Washington, Pa., a drilling contractor of some 30 years' experience in the actual production of oil, who had made a number of improvements in the art of drilling. He had patented the bolted wooden derrick shown in patents No. 852,486, of May 7, 1907, and No. 877,624, of Jan. 28, 1908, to which were added subsequently No. 898,027,

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of Sept. 6, 1908, and No. 946,815, of Jan. 18, 1910. The essential feature of these patents so far as they applied to derrick construction was the use of slots formed on the upper and lower edges of connection plates and braces, instead of round holes, the girts and braces being adapted to be clamped together at the different sections by bolts.

The portable steel derricks manufactured in accordance with these ideas have been known as the Yorke portable steel derricks, advertised as the only derricks which can be dismantled without removing a bolt—which is strictly true of the main joints only.

Inasmuch as the strength of the joints in a structure of this fashion is determined not by the shearing resistance of the bolts but by the friction of the gusset plates against the main members, when the nuts have

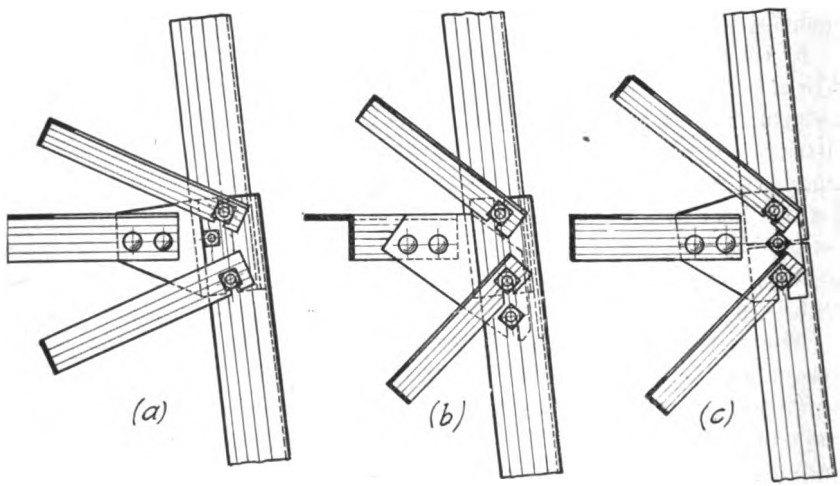

FIG. 37.—YORKE DERRICK JOINTS.

been drawn up tightly, the idea did not appeal very strongly at first glance to men experienced in the fabrication of structural steel. The Carnegie Steel Co., however, undertook the manufacture of the Yorke derrick under arrangements with Mr. Yorke and continued to manufacture it down to the end of 1913, when the association ceased by mutual consent. Mr. Yorke's experience in the actual drilling of wells combined with the experience of the Carnegie Steel Co. in the manufacture of structural steel resulted in the development of a drilling structure well suited for the lighter kinds of work done in standard-rig drilling practice.

The first Yorke derrick manufactured by the Carnegie Steel Co. was shipped Jan. 8, 1909, to the Manufacturers Light & Heat Co. for use near Waynesburg, Pa. It had no steel crown block or steel base, and the ladder was made of wood. Steel bases, ladders, and crown blocks were, however, added in the development of the derrick, and other improve-

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ments were made from time to time, so that their manufacture was ultimately standardized and they were used interchangeably with the Woodworth derricks and made from the same shop detail drawings so as to be interchangeable with them in the construction of complete rigs.

In the first derricks the joints were made as shown in *a*, Fig. 37. The connection plates were riveted to the girts and the derrick legs were overlapped on each other the full depth of the gusset, to form the splices. In May, 1909, details were modified as shown in *b*, Fig. 37, so as to provide a longer splice with a somewhat simpler method of erection, by reason of the elimination of the countersunk bolt at the center of the gusset plate.

In August of that year, the detail shown in *c*, Fig. 37, was adopted.

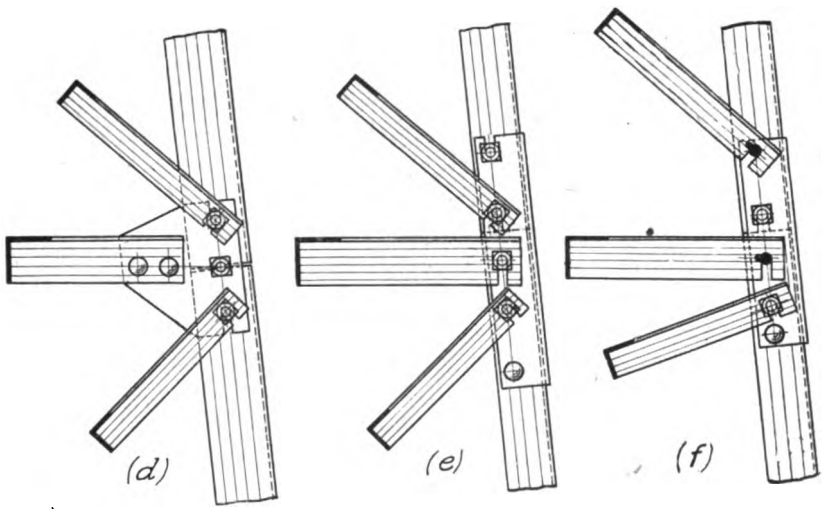

FIG. 38.—YORKE DERRICK JOINTS.

The legs were made abutting instead of overlapping, with the girt riveted to the gusset plate.

This method of construction was followed for two years, during which time a number of them were placed in service. There were, however, one or two failures by reason of the pulling out of the derrick legs from the splices, under the influence of unequally distributed loading on the crown block. The difficulty was finally traced to the fact that the splices were of insufficient length and in consequence a modification was made as shown in *d*, Fig. 38, which is the same as *c*, Fig. 37, except that the splice was made longer. The slots in the diagonals were now made at right angles instead of vertical.

This solved that problem and there were no further difficulties on that score; but when the Woodworth derrick joints were changed from gusset plates to square-root angles in August, 1912, the Yorke derrick joints were modified in the same manner and in consequence were made in

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accordance with *e*, Fig. 38, and so continued to be made until the end of 1913. On Mar. 24, 1914, Mr. Yorke was granted patent No. 1,090,950 covering further improvements in metallic derricks for use in oil, gas, salt, and artesian wells, although equally applicable for mining shafts, hoists, and other towers generally, the parts of which were said to be so constructed and arranged that they might be quickly assembled and disassembled without putting in or taking out a single bolt. The patent lock-bolt construction in its latest form is shown in *f*, Fig. 38, and I understand that a number of them have been manufactured by the O'Brien Steel Construction Co., Washington, Pa., and placed in service in the Appalachian oil and gas fields.

d. The Foukes Joint.—While the Yorke and Woodworth drilling derricks have given good service, and their erection has been simplified and cheapened by a number of improvements, introduced as already stated, the derricks still remain subject to the usual methods of shop

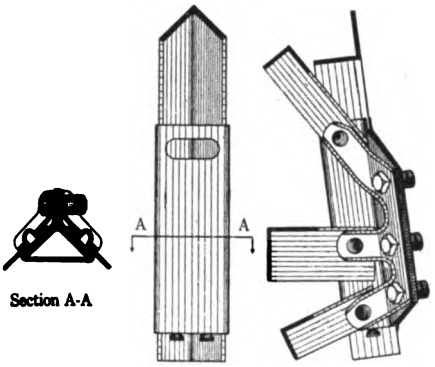

FIG. 39.—FOUKES DERRICK JOINT.

manufacture, whereby wooden templates are made, the holes marked off in accordance therewith and punched. To do away with these elements of shop cost, as well as to simplify the work of erection, various expedients have been conceived, none of which have come to actual fruition with the exception of that now to be noted.

The Foukes joint is the invention of Philip Foukes, Superintendent of the Upper Union Mills fitting shop of the Carnegie Steel Co., and his method of construction is shown in Fig. 39. It consists essentially of a forged-steel plate formed to shape under dies, to take the derrick legs, with an outstanding wing to receive the girts and braces. The derrick legs do not require any punching, except that a small button is formed on their upper ends just below the clamp, which serves as a temporary support during the process of erection. When the derrick is erected the leg section is hoisted into position with the clamp in place, girts and braces are inserted, and, when all connections at the joint have been made, the

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bolts are tightened, and, by reason of the spring action in the plates, not only are they rigidly held in place but also the clamp is firmly attached to the legs.

This joint, tested previously in the shop, was first introduced in an experimental way by the Carnegie Natural Gas Co., in 1914. Seven derricks for drillers and pumpers, 80 ft. and 64 ft. in height, have been erected and have given complete satisfaction.

VIII. Structural Steel Drilling Rigs

By the substitution of structural steel for wood in derrick construction little reduction is made in weight, for the reason that there is little excess of material in the wooden derricks—a result of long experience in their construction. This is not the case, however, with such pieces as samson posts, jack posts, foundation sills and other heavier portions of the substructure. With steel, it is practicable to adjust very nicely to exact requirements the strength, and with it the weight, of members. While some economies have been effected by the use of steel derricks, much more has been gained by the substitution of steel for wood throughout the entire drilling structure, so that, while a steel derrick of equivalent height and strength to a wooden one would weigh approximately the same, a complete steel drilling rig will weigh only about three-fourths as much as the corresponding structure built in wood. This reduction in weight not only reduces the cost but is an important consideration in a structure intended to be semi-portable. For this reason, when the introduction of steel drilling structures was commercially undertaken, endeavors were made to carry the substitution of steel for wood to the utmost extent.

1. *Rigs Wholly or Chiefly of Structural Steel*

It has been noted that the two 72-ft. California steel derricks shipped to South America in 1908 were complete with base, steel walking beam, machinery supports, house framing, and ladder. This was good so far as it went. The next step was to build a rig complete with derrick, crown block, ladders, base, machinery supports, samson post, jack posts, knuckle posts, walking beam, bull wheel, band wheel, house framing, etc., all of steel, and to cover it with corrugated steel roofing and siding. This step was taken in 1908 when four 80-ft. rigs were built for the Carnegie Natural Gas Co. for use in Washington and Westmoreland Counties, Pa., and Wetzel County, W. Va. The construction of the derricks for these rigs has already been referred to. The working parts were built entirely of structural steel, with the exception of the crown pulleys and sand reel, which, having been made already in metal, it was not necessary to redesign. The house framing, however, was continued out only as far as

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the sand reel, the remainder of the belt house and the engine house being of wood (see Fig. 40).

In the same year a complete design was worked out for a 55-ft. rig to be used in cleaning and pumping old wells and for shallow drilling. Rigs of this sort have not been built. The wheels and walking beam of the standard rig have, however, been used with 64-ft. derricks.

Three 80-ft. Woodworth drilling rigs were built for the Carnegie

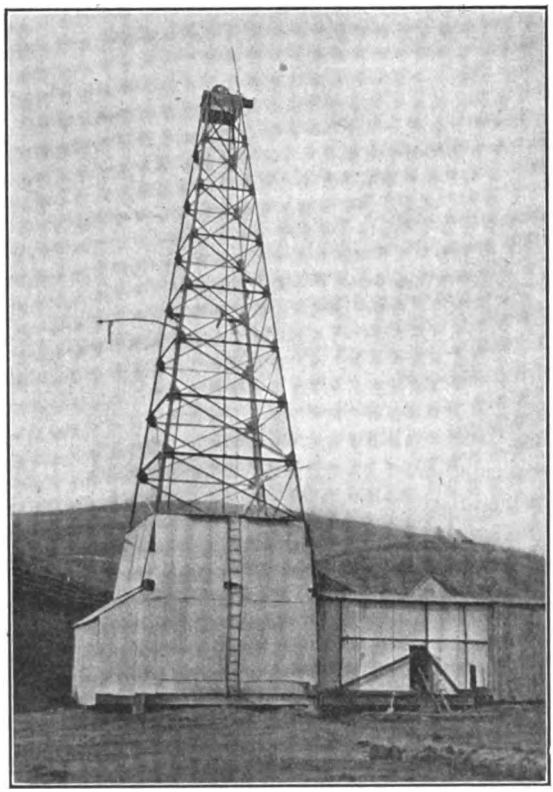

FIG. 40.—80-FT. STEEL DRILLING RIG, 1908. CARNEGIE NATURAL GAS CO., EXPORT, PA.

Natural Gas Co. in 1909 and were first erected at Avon, Ohio, and Lumberport, W. Va., for the drilling of gas wells. It is usually impracticable to follow up the records of drilling equipment manufactured and shipped broadcast to the ends of the earth. In the case of these rigs, however, records of the Carnegie Natural Gas Co. were available. By the end of 1910, one of the 80-ft. rigs made in 1908 had drilled five wells with a total depth of 13,901 ft. and up to the end of 1912 one of the 1909 rigs erected at Avon, Ohio, had drilled six wells with a total depth of

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17,808 ft. One of these wells, C. M. Green No. 1, Avon Township, Lorain County, Ohio, reached the depth of 4,440 ft.

Six 72-ft. California rigs were made in 1909 and shipped to the Burmah Oil Co., Ltd., at Rangoon. These rigs, however, while fitted up complete with derricks, machinery supports, house framing, walking beam and samson post, and metal sand reel, utilized wooden bull, calf, and band wheels, which is an instance of what has happened all through the course of the development. The equipment, while designed primarily to be used as a unit, is interchangeable with other types of material, such as wood and pipe. This would not be the case had the designer not deliberately

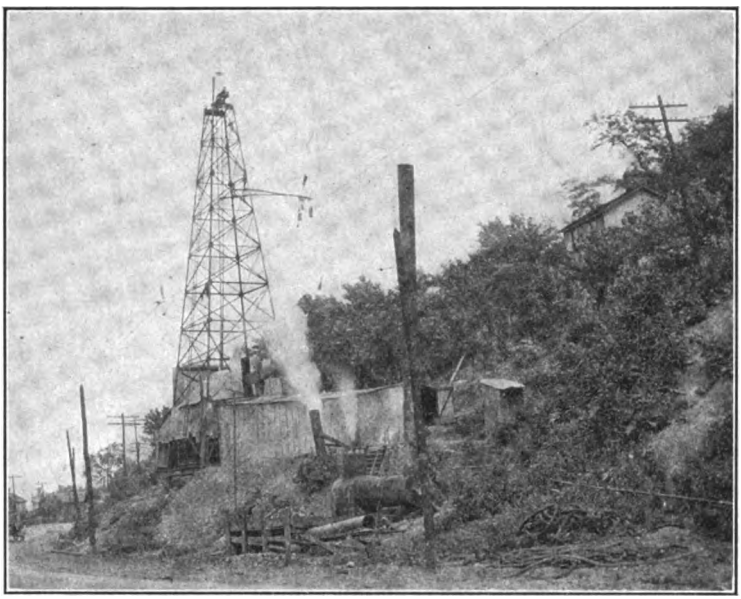

FIG. 41.—72-FT. STANDARD STEEL DRILLING RIG. CARNEGIE NATURAL GAS CO., MUNHALL, PA. RECORD, FIVE WELLS.

chosen to follow the well-recognized lines of oil-well structures and to depart therefrom only for very good and sufficient considerations.

The first standard combination rigs were manufactured in 1912, 80 ft. high, and were shipped Apr. 25. The first went to I. N. Knapp, Houma, La., and the second to Icacos Development Co., Trinidad. In the Knapp rig, it may be noted, the engine was placed between the draw works and the band wheel, both of which it drove by sprocket drive. Mr. Knapp drilled three wells with this rig in the search for oil in Terre Bonne Parish, and after the endeavor was definitely abandoned on account of geological conditions, the rig was sold to the Southwestern Gas & Electric Co., of Shreveport, La., at a very handsome figure with previous use

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considered, and has doubtless drilled a number of wells since. The Icacos rig drilled a fine oil well at its first location in 62 ft. of sand, at a depth of 2,859 ft., the first 1,032 ft. of which was drilled with cable tools and the remainder with rotary, the diameter of the hole being 16 in. at the top and 8 in. at the bottom.

The first 80-ft. California combination rigs were built the same year, three of them going to the Penn-Mex Fuel Co. at Tampico. The first 80-ft. heavy California combination rigs were made in 1913 for the Carib-

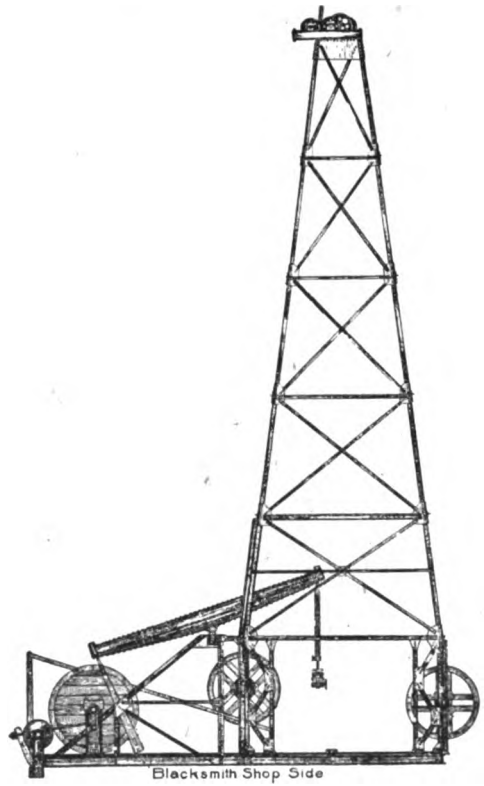

FIG. 42.—72-FT. WOODWORTH CALIFORNIA RIG. THE 1904 STRUCTURE.

bean Petroleum Co., and shipped to Maracaibo, Venezuela. The first 86-ft. heavy California rig was made in 1913 on order of the Oil Well Supply Co. for the Calgary Petroleum Products Co., Calgary, Alberta, Canada.

It has been impracticable to follow up the record of these rigs. They have been built on the same general lines with California double-drum sand reels located on the walk side. In 1913, however, a rig was built for the Hope Natural Gas Co. for use in West Virginia to take a very long special sand reel set on a bevel. This, of course, necessitated a

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thorough revision of the general design. While some drillers have objected somewhat to the use of the straight-line reel set on the walk side, experience has indicated that drilling rigs can be built much more compactly in that way; and when standard rig drillers become accustomed

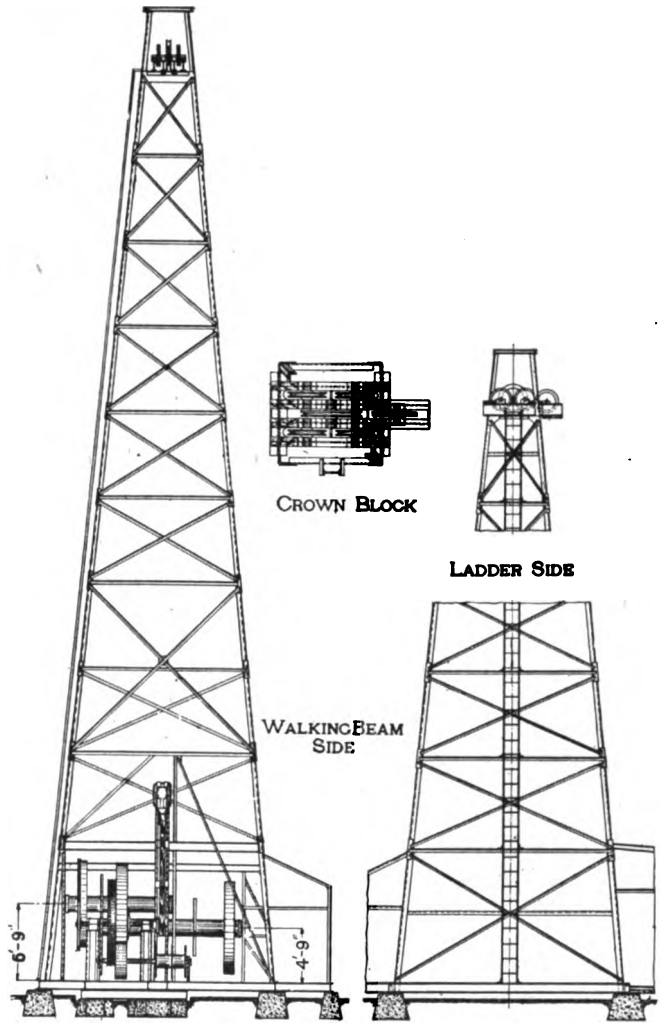

FIG. 43A.—80-FT. WOODWORTH CALIFORNIA DRILLING RIG. THE COMPLETE MODERN STRUCTURE.

to the modification by practice, the construction is found to be very satisfactory; just as in the case of the wooden California rig, in which the use of an iron sand reel set on the walk side has been standard practice for many years (see Figs. 43A and 43B).

Practice in the construction of combination rigs is not yet absolutely

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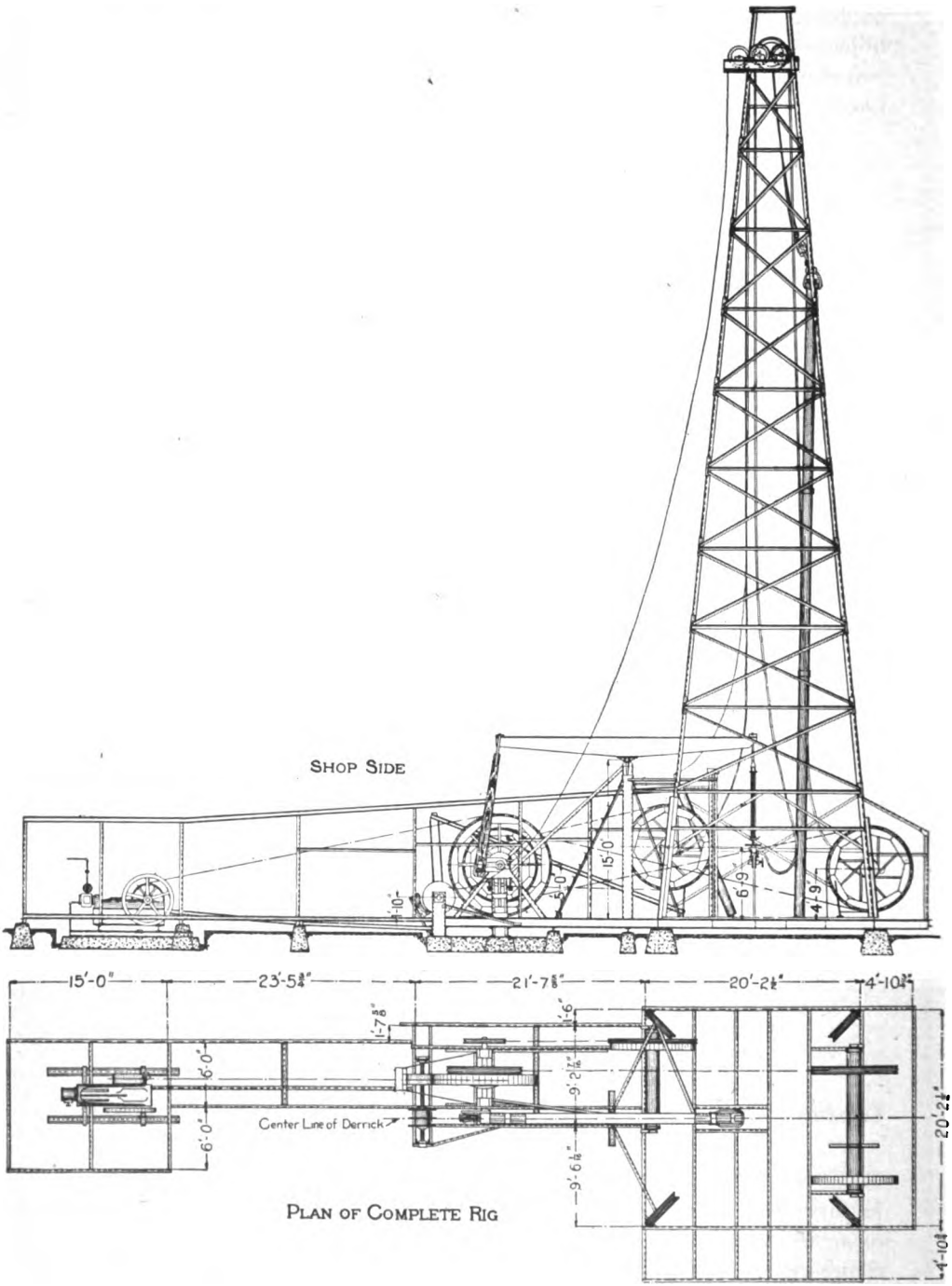

FIG. 43B.—80-FT. WOODWORTH CALIFORNIA DRILLING RIG. THE COMPLETE MODERN STRUCTURE.

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standardized. The rigs already referred to have been built for use with one engine, from whose pulley power is transmitted to the band wheel by belt, and from the band wheel to the bull wheel or calf wheel in the usual manner, and to the draw works located on the walking-beam side by supplementary sprocket on the band-wheel shaft. The 106-ft. heavy California combination drilling rig (Fig. 44) now on exhibition at the

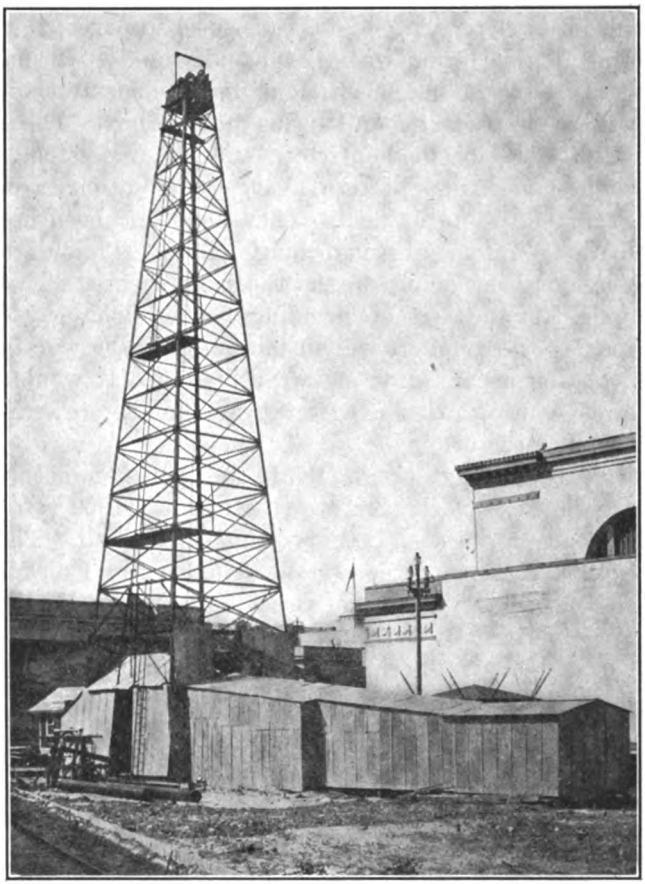

FIG. 44.—106-FT. CALIFORNIA COMBINATION DRILLING RIG. PANAMA-PACIFIC INTERNATIONAL EXPOSITION, 1915.

Panama-Pacific International Exposition may be considered as representing the culmination in this line of development and also as the most complete drilling machine ever constructed. It is designed for use with two engines, one to drive the rotary and the other for use with cable tools. The walking beam is mounted on revolving center irons, and can thus be swung out of the way of the drill pipes when the rotary is in use. As a consequence, the most rapid interchange of drilling tools can be effected,

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and drilling proceeds with rotary or cable at the option of the operator. All the material is structural steel except the floor platforms, the cants for band, bull, and calf wheels and the nailing strips to which the corrugated iron covering is attached. The bull wheel is 8 ft. in diameter with an 18-in. O. D. pipe shaft, 16 ft. long over gudgeons: the calf wheel is 7.5 ft. in diameter, 16-in. pipe shaft, 90-in. sprocket; the band wheel is 11 ft. in diameter, 12-in. face, with 7-ft. tug wheel, double drive; the band-wheel shaft is 6 in. in diameter with 42-in. calf-wheel sprocket; the sand reel is a California double drum, with 42-in. pulley and 40-in. flanges; the walking beam 12 by 24 in., 26 ft. long between centers of bearings, equipped with cable protector and spring pitman box. The pitman is of wood. This could be made of steel, and it is understood that steel pitmans (Smith and Hess pattern) have come into recent use on wooden rigs in the California field. They have also been used experimentally in West Virginia gas-well drilling. The steel foundation of this rig is set on concrete blocks to represent the most recent and best practice. The total weight of the rig, not including wooden floor, platforms, and nailing strips, is 81,310 lb. It will sustain an ordinary working load of 223,000 lb. on the usual factor of safety of four. It would probably require a direct pull of at least twice this amount before serious injury were done to the structure.

Structural steel drilling rigs adapted to the Canadian method of drilling with steel derricks 56 ft. high are shown in the 1909 catalog of the Deep Well Tool & Boring Co., Ltd., St. Albans, England. They appear also in the 1912 catalog of the Oil Well Engineering Co., Manchester and London, England. Steel drilling rigs adapted to the Galician method of drilling are advertised by the Galician-Carpathian Petroleum Corporation. The writer has no specific information as to how far these rigs have come into use, nor details of their construction.

2. *Pipe Rigs*

As already noted, the principle of interchangeability has been kept in mind in this development. Parts of the steel equipment may be, if desired, of wood. Such use does not concern the manufacturer. He is interested, however, when structural steel drilling equipment is to be used with pipe derricks, since in that case he must work out the shop details necessary to insure satisfactory operation.

The first structural steel rigs for use with pipe derricks were made in 1909 for the South Penn Oil Co. Eleven of them were used in that year in the Shinnston district. These were not complete. What the structural steel man furnished was the derrick foundations, machinery supports, house framing, corrugated sheeting, walking beams, and bull wheels. Pipe derricks were made by the South Penn Oil Co. in its own shops, and

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the band wheels, sand reels, band-wheel shafts, etc., were taken from its own stock. Numbers of these rigs, more or less complete, for use with pipe derricks, have been built since that date, chiefly in conjunction with Lee C. Moore & Co., who are the manufacturers of the Neill pipe derrick, as already noted. The South Penn Oil Co. has some three dozen of these

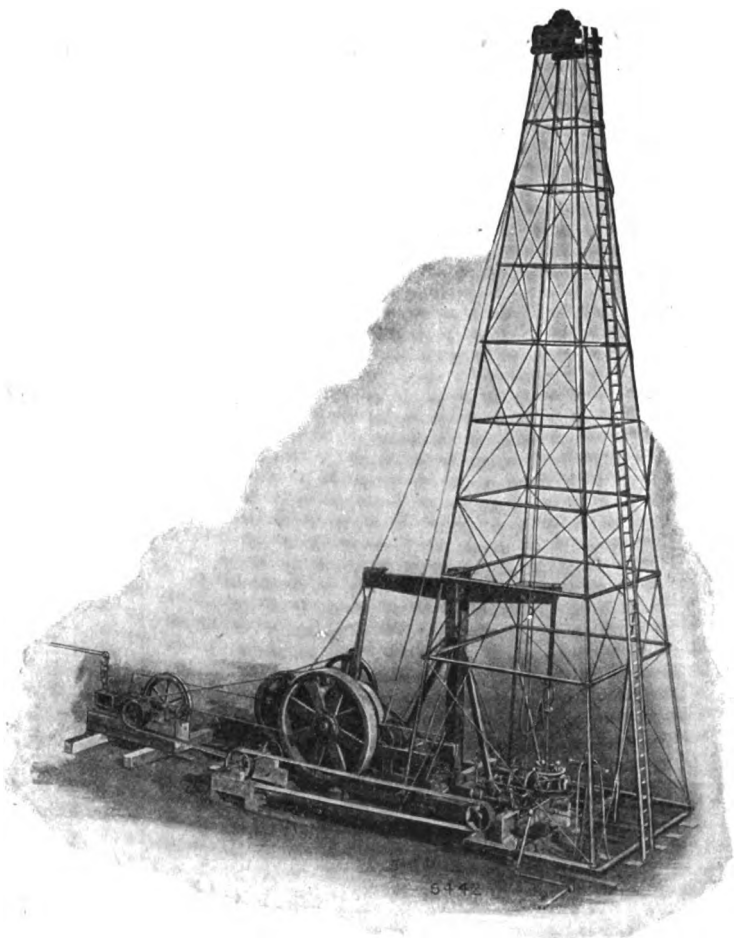

FIG. 45.—STEEL-PIPE DRILLING RIG FOR CABLE DRILLING. INGERSOLL-RAND CO. NEW YORK, N. Y.

rigs in use in its extensive drilling work in West Virginia. The very heaviest so far built, however, were two 95-ft. California combination rigs constructed in 1914 for Lee C. Moore & Co. and shipped to the Anglo-Mexican Petroleum Products Co., Tampico, Mexico. These rigs weighed 53,000 lb. each, not including the pipe derrick.

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3. *Parts of Rigs*

Just as improvements have been introduced in the manufacture of steel derricks from time to time, generally in the direction of the simplification of details and the reduction of weight, so also have improvements been made in the separate parts of the drilling rig, which are worthy of notice as bearing on the present state of the art and the methods by which it was reached. Some of these are covered by letters patent: No. 932,081

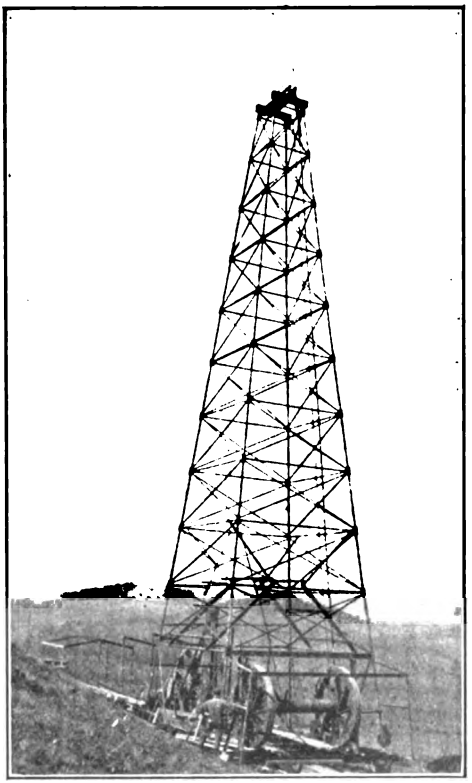

FIG. 46.—STEEL-PIPE DRILLING RIG FOR OIL-WELL DRILLING. SOUTH PENN OIL CO., BLUE CREEK, W. VA.

and No. 1,090,184 on bull wheels, No. 1,020,069 on band-wheel construction and No. 960,474 on the general assembling and make-up of a complete drilling rig.

a. Crown Blocks.—The first two-pulley crown blocks were constructed of steel channels with babbitted top and bottom bearings, and used on derricks made in 1908 and 1909. The users, however, did not seem to require top bearings and so these were discontinued. As a matter of fact, a great many still prefer not to have any metal bearings whatever,

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but to use pieces of wood. The first crown blocks were without any protection against cable wear. The present and most approved method in the construction of two-pulley crown blocks for standard drilling is to make them of steel beams with babbitted bottom bearings and to line the top and bottom flanges with wood, in order (1) to prevent the drilling cables from coming in contact with the sharp edges of the beams at the bottom; and (2) to provide a proper wearing surface at the top where sheaves are hung below the crown block for pulling the casing (see Fig. 47).

The four and six pulley California crown blocks are likewise made of beams and protected by wood; but inasmuch as these blocks are used with wire lines, the most recent tendency is to omit wood, which is likewise done in the case of the five-pulley block used on rotary derricks. The 106-ft. steel drilling rig at the Panama-Pacific International Exposition is furnished with a seven-pulley crown block with babbitted bottom bearings, and with top steel bent plate covers fitted with grease cups.

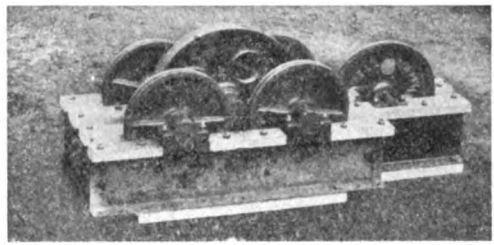

FIG. 47.—SIX-PULLEY CALIFORNIA CROWN BLOCK.

The steel crown block has been in use many years on wooden derricks. The first known to the writer were used in Texas on rotary derricks in 1909. Very heavy blocks fitted with special bearings, self-oiling devices, etc., have within the last year or two come into use in the California fields, and are made by the larger dealers in oil-well supplies in their own shops. Some of them are made to permit adjustment in the position of the pulleys at almost any desired location, extra holes being provided for that purpose. Such are used by the Standard Oil Co. of California in its rotary drilling.

b. Bull Wheels and Calf Wheels.—Steel drilling rigs are characterized by the use of bull wheels built with channel spokes and wood-filled plate rims attached to a pipe shaft. The spokes of the first bull wheels made in 1908 were constructed of 6-in. I-beams connected to the shaft through forged angle flanges and set radially, and the rims were made of steel channels with the flanges turned outward and with wooden cants laid in and bolted thereto in the line of their depth. The gudgeons were of bowl type and were screwed on the outside of the pipe with stand-

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ard threading with, in addition, four set screws to prevent slipping (see Fig. 48).

The wheels are now made with 16 or 18 in. O. D. pipe shafts and with inside gudgeons bolted to the pipe. The spokes are made of steel channels, bolted tangentially to the shaft, which method does away with the forging required in making the angle flanges. The brake wheel has a 9-in. face and is made with a plate rim riveted to the channel spokes and carrying wooden fillers similar to the standard wooden cants. The tug wheel is made in the same way, except, of course, that the cants are grooved for double tug, and provision is made for the attachment of a wooden dog for drawing the rope on and off. The hand holds for turning the wheel are made of gas pipe bolted by ⅝-in. bolts through the entire thickness of the wooden cants and the plate rim.

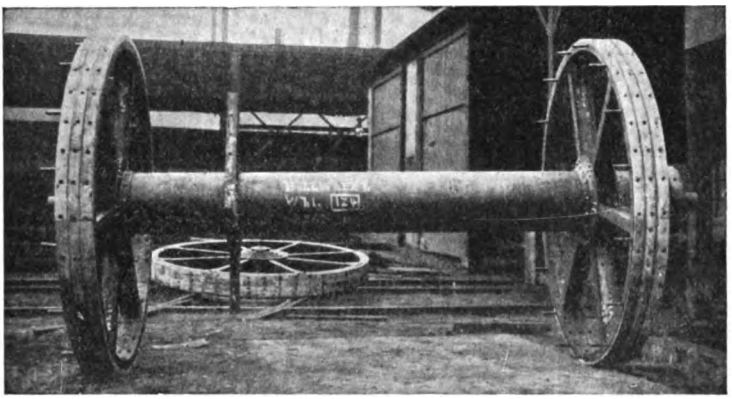

FIG. 48.—8-FT. STEEL BULL WHEEL, 1908. RIMS ARE NOW MADE AS SHOWN IN FIG. 50.

The spool is made of steel angles bolted to the pipe shaft and connected to each other by gusset plates so as to form practically a single piece. Protection of the drilling cable from the sharp edges of the steel is given by two coils of rope wrapped around the shaft next to the tug wheel (as is often done in a wood wheel) or by wooden strips bolted to the spokes, the latter being the preferable manner. Otherwise, the wheel is ready for use as it comes from the maker.

Bull wheels are usually shipped in four pieces, the two wheels, spool, and shaft being separate. Patent No. 1,090,184 discloses a sectional wheel designed with a view to facilitate shipment. No wheels made in accordance therewith have yet been constructed. Bull wheels with metal tug rims, however, have been made for experimental purposes and their service is now under observation.

Some other types of bull wheels may be noted in passing. Ross and Seeley, Hollywood, Cal., patented Jan. 10, 1911, No. 981,128, iron bull-

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wheel and calf-wheel shafts, made of iron pipe with outside screwed gudgeons. The rims and spokes are of wood and connected to the shafting by heavy castings (hubs). The calf-wheel shafts are lagged with rope to make the shaft the proper size and form a cushion for the wire casing lines. The bull wheels are not lagged, and the spools are constructed by the use of heavy wood friction blocks made of 6 by 16 in. timbers. Special bearing boxes are provided for attachment to the wooden bull and calf wheel posts. A number of these have come into use in California.

Within the last year or two, the California fields have seen the introduction of the Keck and Hively bull wheels and calf wheels. These consist of pipe shafts with bowl gudgeons and square wooden friction blocks bolted to the shaft for the attachment of the wooden spokes necessary to complete the rim. Otherwise, it is a device for utilizing on

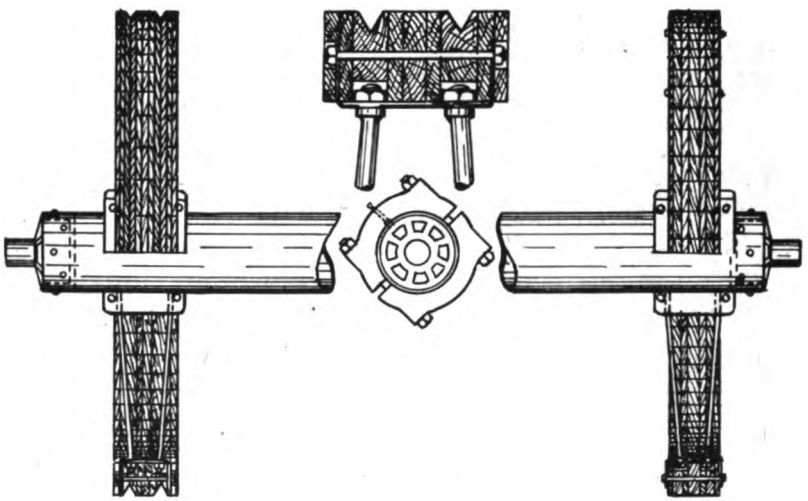

FIG. 49.—MOORE SECTIONAL BULL WHEEL.

a metal shaft regular wooden cants and spokes customary in the construction of wooden rigs.

Moore's sectional steel bull wheels, however, a few of which have been introduced in the Eastern oil and gas fields, are designed to provide an all-metal wheel with the exception of the rim. They utilize 16-in. O. D. pipe, inside cast-steel gudgeons, sectional hub castings, and tangent screwed spokes, carrying at their circumference metal plates on which the cants are set, as distinguished from the method employed in the Woodworth wheels, in which the cants are set within the rim (see Fig. 49).

Woodworth calf wheels have been made in the same manner as the bull wheels and indeed with the same size of pipe and same gudgeons, with a view to the most economic manufacture. The first wheels made in 1909 were without flanges; but flanges are employed in most recent construc-

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tion. The wheels have been made with wooden grooved rims for manila rope drive, with iron rims for manila rope or wire line drive and with sprocket rims for sprocket drive.

c. Band Wheels.—The band wheels furnished with the rigs of 1908 were 11 ft. in diameter, with I-beam spokes, channel rims, and the regular cast band-wheel flanges keyed on a 4½-in. band-wheel shaft. The flanges of the channel rim were turned inward, and a canvas belt was fastened to the outside, so as to provide proper friction against the sand-reel pulley. The cast flanges proved too weak for this service, and on subsequent wheels were replaced with wrought-steel hubs built up of successive plates. The flanges on the channel rim were then turned outward, and filled with wooden cants similar to those first employed on the steel bull wheel, which

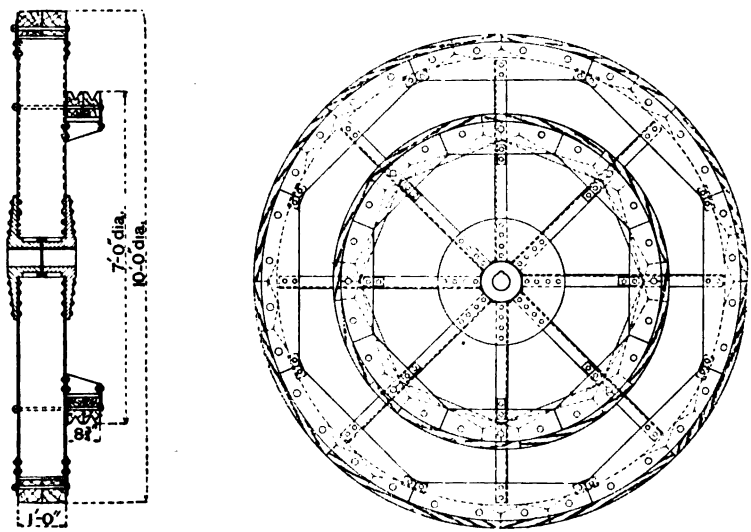

FIG. 50.—10-FT. STEEL BAND WHEEL, 1911. THE RIMS OF THE STEEL BULL WHEELS ARE NOW MADE IN THE SAME MANNER.

gave better friction against the iron sand-reel pulley. In both cases the tug wheel was made of a channel rim, filled with wooden cants and connected with the spokes of the band wheel by angle brackets. The present type of steel band wheel was adopted in 1911. The hubs are made of two solid rolled-steel flanges, abutting at the center of the wheel, with channel spokes and plate rims, within which are secured the wooden cants. The tug rim is made of grooved cants, faced with a plate and supported on brackets. A hub of this character was first used on a band wheel by Patrick Yorke early in 1911, but the construction of the present wheel deviates very widely from that original form; and the manufacture of the hubs by rolling instead of casting is only possible on the mill operated by the Carnegie Steel Co. at Homestead Steel Works in

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the manufacture of a very great variety of annular sections such as car wheels, crane-track wheels, locomotive pistons, etc. The band wheel was first made 11 ft. in diameter, but experience in their use indicates that a 10-ft. wheel on a steel rig is as effective a prime mover as an 11-ft. wheel on a wooden rig (see Fig. 50).

d. Walking Beam.—For some reason or other, designers of steel walking beams have considered it their duty to depart rather widely from the ordinary wooden construction which so admirably served its purpose

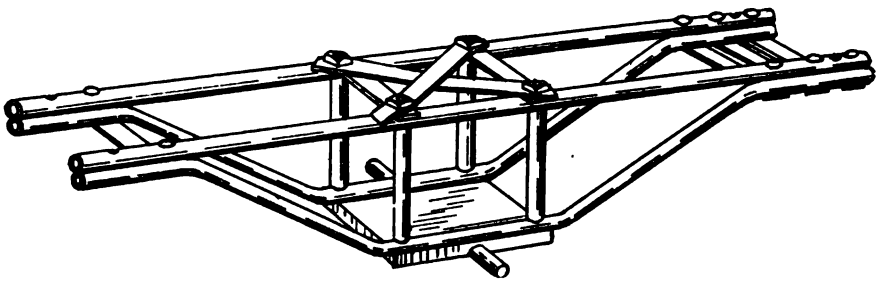

FIG. 51.—SHANNON PIPE WALKING BEAM.

all these years. This has been done by the use of truss rods over the top of a light wooden beam, as was formerly done in the construction of underframes for steam-railway freight cars. Such a reinforced walking beam appears even so late as the 1914 edition of the catalog of a large dealer in oil-well supplies. The attempt was next made to use pipe. Patent No. 914,608, issued to Hezekiah Shannon, of Cyclone, Pa., Mar. 9, 1909, shows an all-metal walking beam made of bent pipe and steel bars. The writer does not know of any actual use of a beam built in accordance

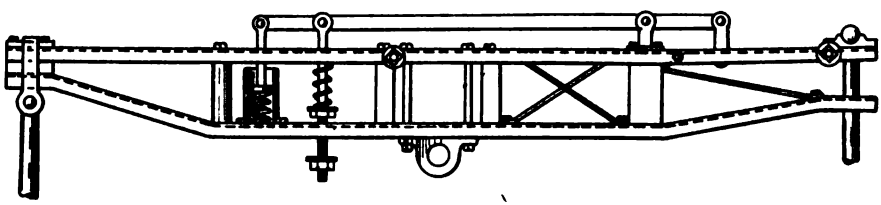

FIG. 52.—LONGTINE STRUCTURAL WALKING BEAM.

with this particular patent but has seen similar beams in actual use, one in the McDonald oil field in the year 1909 (see Fig. 51).

He also saw in February, this year, at Brea, Cal., a very ingenious affair built up with toggle joints, springs, screw adjustments, etc., by the Los Angeles factory of the Oil Well Supply Co. for one of their customers. He made no sketch of the structure at the time, but the beam was similar, if not identical, with that patented by Robert B. Longtine of Taft, Cal., No. 1,009,537, Nov. 21, 1911 (see Fig. 52).

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The beams built by the writer have been exact equivalents of the standard wooden walking beams. They are made in box-girder fashion of steel plates and angles, customary tapered shape, and with corresponding notches for temper screws. The first beams were made with wooden bearings and without any cable protector on the temper-screw end. The improvements introduced since 1908 are chiefly a bent pipe at the temper-screw end for a cable protector, and babbitted cast bearings for the pitman. The beams are designed for support on a standard cast-iron saddle, such as is carried in stock by dealers in oil-well supplies, and are interchangeable on wooden or steel rigs. Combination rigs have been equipped with the modern revolving center irons, as already noted.

A great number of steel walking beams are in use. There has been but one real objection to them, and that is that their stiffness made it more difficult for the driller to determine exactly what was going on at the bottom of the well. This stiffness is in part due to the fact that while the steel beams are designed to be of the exact theoretical strength of the standard wooden beams, for which they are substitutes and are very much lighter, yet owing to the greater homogeneity of steel they are really much stronger than the wooden beams. Some three years ago the Oil Well Supply Co., at the instance of the South Penn Oil Co., devised a special spring pitman box, which has been utilized on steel beams and on wooden beams as well.

Steel walking beams with single webs, built up plate-girder fashion (I-shape), have also been devised, but have not come into anything more than limited use, the preference of the driller for what he has been accustomed to use being somewhat greater than his desire to save slight additional expense.

In the first samson-post design, made by the writer as far back as 1904, he contemplated the support of the steel walking beam on a pin driven through its center exactly as is done in other operating mechanism, and this method of construction is advocated in recent publications. It did not come into use, however, until the present year, when it was employed on a heavy California drilling rig built for C. W. Washburne, Angola, West Africa.

Conclusion

While it is not at all likely that the steel drilling rig will replace the ordinary wooden structure to anything more than a limited extent, particularly in the drilling of wells for oil, a sufficient number have been put into use to justify conclusions as to their advantages, based not on theoretical considerations but on actual experience.

The steel shop methods insure that if the structure goes together at all, it will go together with a precision not possible in wooden construction. If the sills are level, the structure will be plumb and square without

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any particular attention on the part of the rig builder to make it so, and simply for the reason that it has been made so in the shop. It is a question, therefore, whether the steel structure should not rather be called a drilling machine than a rig, and indeed that method of designation was used in the first pamphlets issued by the Carnegie Steel Co. It was withdrawn for the reason that the word "rig," while designating in ordinary parlance something more or less crude, is a standard term without reproach in drilling practice.

The necessary consequence of this precision in manufacture is seen in the smoothness with which all parts of the mechanism work. The structure is stiff and rigid. It does not give like a wooden rig and in consequence there is no lost motion in operation. The thing which was particularly noted by P. Keegan, who operated a 72-ft. drilling rig at Ladybrand, Orange Free State, South Africa, in 1910 on the property of the South African Oil Co., was that he only needed to carry 80 lb. of steam on his boiler. His experience in the reduction of boiler load has been amply corroborated by others. The difference between the pressure necessary on a Woodworth standard drilling rig and a crude wooden structure is due to the smoothness of operation resulting from precision in manufacture and alignment.

In 1892 when the Oil Well Supply Co. first described the steel derrick in its catalog, the theoretical statement was made that while the first cost would be more than that of a wooden rig, it would be amply compensated for after the completion of the third well. It is reported that some of the steel derricks have been used 20 or more times. Records have been kept of the use of particular steel drilling rigs and they show more than a half a dozen wells each. This argument is, therefore, a good one. At present, a steel drilling rig in the Eastern oil and gas fields will cost, erected, about 1¾ times as much as a wooden one, but it represents an investment of indefinite use and practical indestructibility. Assuming that a wooden rig will drill two wells, and a steel rig, say, only four or five, the investment in the latter, capitalized at the usual rate of interest, will yield larger returns, by reason of its small cost of maintenance. This is, of course, particularly true in the case of the very high and powerful structures which deep well drilling has recently made necessary.

In addition to these considerations may be mentioned briefly the fireproof character of the structure, its small wind exposure, and the ease of its erection. Not long since, a steel rig was struck by lightning, which only destroyed a few wooden planks at the top and some flooring. Another rig suffered not long since from a gas explosion which destroyed the cants from the bull wheel and a portion of the wooden floor, but did absolutely no other damage. The surface of the derrick proper exposed to wind is not more than half as great as that of the wooden structure of the same height, strength, and stiffness.

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Discussion

R. B. Woodworth, Pittsburgh, Pa.—This is a very long paper. It is a structural engineer's contribution to the art of mining and is written from his standpoint rather than from that of the mining engineer, and is, therefore, a study in the development of drilling structures by one who has made it his business to design structures rather than mechanism and machinery and rather than to utilize those structures in the actual exploitation of oil and gas.

Drilling practice varies widely in different oil and gas fields. The problems of western Pennsylvania are much simpler than the problems of the Mid-Continent and of the Gulf Coastal Plain and these again are simpler than problems of the Pacific Coast.

The history of the commercial development of oil and gas is recorded in the annual publications of the U. S. Geological Survey and other bodies whose business it is to make record of statistics. The mechanical and structural features of the art of drilling have not yet received adequate treatment by the historian. Scattered references may be found on the subject in almost any work dealing at large with petroleum and its production. These references, however, are to those particular aspects of the matter that were contemporary with the publications themselves, but so far as I am aware there is no continuous and comprehensive account written of the development of drilling structures from their beginnings down to the present time, and certainly the more modern application of steel in the construction of such structures has not previously had an adequate discussion.

With this latter phase of the subject I have been intimately associated, indeed, taking to myself the credit for having, in a way, been responsible more than any one else in the United States for the application of steel in the manufacture of drilling structures, and the purpose of the paper which is now before you is to record particularly the steps in that development while the subject is still fresh and the data available.

In the very nature of the case, the discussion has been limited to structures originating in the United States to the exclusion of well-known types of drilling methods and their corresponding structures which are in use in Galicia, Russia, and elsewhere. While particular attention is paid to the use of steel, at the suggestion of Captain Lucas I have embodied in the paper a brief discussion of the various stages which have been followed in the evolution of drilling structures from the early days of the salt wells down to the present time. Though brief, I hope that this discussion and the paper itself as a whole will form a basis on which may be built some day that comprehensive account of the evolution of drilling methods and structures which is the desideratum of engineers and operators alike and which, when it is written, will be a record of

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the application of inventive genius to the solution of the successive problems which the development of the industry has brought forward for consideration.

CHESTER W. WASHBURNE, New York, N. Y.—The question that has come up, of steel derricks *versus* wooden derricks, is very close to my own work, and I would like to get the opinion of those present who have examined the same question.

In the Comodoro Rivadavia district, in the Southern part of South America, wooden derricks were used quite successfully in the beginning but they would stand rebuilding on new locations only once or twice. It was finally decided to use steel derricks, the deciding point being that steamer freight was so much less for steel derricks. That was a dry region, a desert, where wood lasts as long as it can stand the wear and tear.

The second job in which this same problem has come up has been in the Tropics. There we have insects that eat up wood, and very severe weather conditions which rust away the steel, but it appeared that the steel could be preserved by painting, while there is no certain way of preserving wooden derricks.

The remark about the greater precision of steel derricks in alignment and plomb seems to me rather immaterial. Good workmen can build a wooden derrick in perfect alignment and plomb. For long transportation to distant parts of the earth I think the steel derrick has every advantage: it occupies less space in a ship, and therefore it is shipped at very much less expense; it can be taken down and set up repeatedly; properly painted it stands any climate.

I notice in the American fields the steel derrick is not yet in very great favor, and I would like to obtain more information as to the reason for its slow adoption.

I. N. KNAPP, Ardmore, Pa.—Mr. Woodworth's paper will serve as a landmark for those who desire information on the subject, as it authoritatively describes practically all that has been done in America on well-drilling structures from the beginning up to the present day.

Mr. Woodworth designed a combination steel drilling rig for me for deep (3,000 ft.) drilling. I used it with complete satisfaction on three wells and if I had abandoned the steel rig then I would have saved a little money over using wooden derricks for the same purpose, buying lumber at $24 per thousand and supplying the lumber waste in making two moves.

As it was, I sold the rig for 70 per cent. of the original cost, so really the steel derrick was a good investment.

These derricks have much less wind surface than those made of wood so are less liable to be wrecked by storm and also less likely to damage by fire.

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I think the steel derrick will come into general use.

The reason why the steel derrick has not come into use more rapidly in the United States is that those who have sold these derricks have not sent out men to erect them the first time. The ordinary driller and worker around a derrick is a very conservative man, and he does not like to work on a derrick into which he cannot drive nails as he has been used to, until some one has shown him he can work with a steel derrick. The drillers are very good men, but they are not educated.

The reason that we have not more written information about well-drilling methods and structures is because those who have followed up this business have done so from laborers' positions. Very rarely do you find an educated man who can handle the drilling tool efficiently, and on the other hand, engineers, geologists, and other persons who write on drilling subjects, know practically nothing about them. They go out and walk around a derrick once; everything looks simple, and they think they know all about how to drill. A bulletin called Well-Drilling Methods has been published by the Geological Survey. The first cut in it is supposed to represent a spring pole. It is a well sweep. All through this bulletin there are many lamentable errors showing that the author had no practical ideas of the use of the various tools and methods. I do not know how many times I have read descriptions of drilling jars which were absolutely ridiculous. A man can drill better in cable-tool drilling without the jars than with them. Of what use are jars? To jar the bit loose when it sticks and in fishing. In some places in Ohio they do not use jars in the string of drilling tools when drilling. If the drill gets stuck, a bumper is run down on the sand line, and the drilling cable pulled up tight, then bumping on the rope socket jars the bit loose.

SAMUEL S. WYER, Columbus, Ohio.—As a member of the Advisory Board of Engineers working jointly with the United States Bureau of Standards in the preparation of a National Gas Safety Code, I am very much interested in the matter of safety. Can anyone furnish data showing any difference in accidents (I mean ordinary operating accidents), as between steel derricks and wooden derricks? Also, can anyone furnish data comparing the lightning hazards of steel derricks and wooden derricks?

There is one feature in regard to the lightning hazard, in every case where I have inspected a steel rig, which has always been overlooked, namely, that you can make a good lightning rod out of it simply by using a jumper around the wooden foundation, and grounding the jumper adjacent to the structure.

LEONARD WALDO, New York, N. Y.—Some time ago a derrick fell into an excavation in the construction of the subway in this city, and in a subsequent suit for damages brought by the people who were injured

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the question of the reliability of the top derrick hooks came up. It was shown that some forms were vastly safer than others. I cannot recall the details of the case here, but I think a number of photographs taken at the time showed the lines of rupture of steel joints at the upper end of the derrick, such as are shown on p. 243 of this paper.

In regard to the relative safety from lightning charges, the total amount of metal in the steel derrick or on a wooden derrick is very small. The static charge would be relatively small. The difficulty of properly grounding anything in structural engineering is very great. Anybody who has attempted to measure grounds in tank construction for large oil tanks has been amazed at the difficulty of securing adequate ground, and there is not much chance of ordinary structural engineering contractors ever giving the proper grounds for steel derricks, but, on the other hand, the danger from the derrick itself is almost a minimum. The common idea that a piece of metal attracts lightning is erroneous. Lightning is not attracted. Lightning is an exchange of two potentials, and I do not think there would be any sensible difference, as far as danger is concerned, between a steel derrick and an ordinary derrick of wood with steel fittings attached to it.

CHESTER W. WASHBURNE.—I ask Mr. Waldo what he suggests as the most satisfactory method of grounding?

LEONARD WALDO.—I had a friend who occupied a house on a hill. He lost his house in three consecutive lightning discharges, and came to me to ask how to make his next house safe, because he was determined to live at no other place than on that hill. I told him to get several hundred feet of barbed wire fencing, with the projecting points sticking out everywhere so that there would be many hundreds of the points exposed at the ridge poles and other places. Grounding these wires is the greatest difficulty. I told him to take a piece of copper sheet, as wide as he could get it rolled (42 or 48 in.), about 6 ft. long, dig a big hole, and get down below the line of moisture of the sand and the foundations of that house, put one of these plates at each of two alternate corners of the house, carrying the leads down to these places, being careful that there were no soldered joints which could not be inspected. He faithfully carried out this suggestion, and his house has since stood through many vicious thunderstorms through a period of about 15 years. I do not know any better way than that to ground, burying large copper plates at oppositely diagonal points of the house, being sure to get below the dampness.

The lightning conductor or lightning rod has no function for carrying off a charge of lightning. You might as well try to stop a swift avalanche with a picket fence as to stop lightning charges once they are started. The object of a lightning rod is to offer a number of points, to make a

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bypass, so that as the lightning charge or the potential cloud meets there, it eases the pressure between the ground and the clouds, and by diminishing this pressure the liability to discharge is minimized.

I have seen a number of powder magazines which were adequately protected, and never knew of anything being really struck, when it had protection of that nature.

DAVID T. DAY.—It seems to me while we are on the subject of the protection of derricks against strokes of lightning, there is a good deal that can be said. Certainly we are very frequently successful in the matter of protecting derricks against lightning. I recall particularly a steel observation tower at Mount Weather, in Virginia, on top of the Bue Ridge, very much exposed. I remember that the observers used to make it a practice in thunderstorms to go up on top of this tower and get an electrical bath. They reported that there would be a tingling sensation all the time, but it was not too much for them. The construction of the tower was such that while it was very evident that there was a great deal of electricity around it during heavy lightning storms, there were no harmful results to the tower.

I. N. KNAPP.—I never knew of a derrick being struck unless there was some gas escaping. Lightning seems to be peculiarly liable to strike where there is gas flowing from a well or tank. I have seen lightning strike within 500 ft. of a steel derrick, right into the open marsh, and the derrick was not affected in any way.

DAVID T. DAY.—The matter of the introduction of the steel derrick is practically at an interesting stage. It seems to me, in reply to Mr. Washburne's question as to why the steel derrick is not being introduced more rapidly, that the best answer is, that it is. It is, considering all conditions, being introduced quite rapidly. I think there is no question about the coming of the steel derrick. There is one prejudice among these drillers, who have been described as uneducated men; perhaps a better way of expressing it is that these drillers are very highly educated men in certain lines, but their education has not been particularly broadly distributed—they are thoroughly educated in favor of the wooden derrick, and they tell me that one reason why they are prejudiced in that direction is that when they put a strain on the wooden derrick it creaks before it pulls in, and a steel derrick does not creak. In closing this discussion Mr. Woodworth could perhaps describe some method by which they could make the steel derrick creak before it pulled in. I believe it would aid in the development of the steel derrick.

R. B. WOODWORTH.—I will say that in my connection with the manufacture and sale of steel-drilling equipment, two instances have come to my notice where steel derricks have been struck by lightning

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or subjected to fire risk. In the first case, lightning struck the derrick and consumed wooden planks laid down at the crown block and some flooring, but absolutely without injury to the steel structure. Another rig suffered not long since from a gas explosion which consumed the cants of the bull wheel and a portion of the wooden floor in the derrick proper but did absolutely no other damage.

In consideration of the fact that steel derricks are usually found in very exposed situations, it seems to me that the small amount of damage which has been experienced in their actual use indicates that the metal tends to dissipate the lightning rather than to attract it. In any event, the possible damage which might be done is very much less than in the case of wooden structures. A steel derrick set on a metal foundation is after all not a very bad lightning conductor, especially if care is taken to ground the structure below the foundation.

I should say that so far as the use of steel derricks is concerned, while their introduction has been gradual like any other new article of manufacture, it has been satisfactory when consideration is given to the cheapness of wood in most all well-known oil and gas fields. The points which were mentioned by Mr. Washburne are exactly the points which account for the manufacture of the first derricks in the United States. They were shipped to a tropical country where insect ravages were more or less acute and where cheapness of transportation due to lightness in weight was a feature.

The reason why observers do not see more steel derricks (using the word steel to cover both structural steel and pipe) is because of the magnitude of the industry. There are, roughly, 23,000 wells drilled per annum for oil and gas in the United States. It is practically impossible to have all of these drilled with steel equipment, and I think the manufacturer would be perfectly satisfied if his equipment were purchased for 1,000 of these in each year.

C. W. WASHBURNE.—May I ask for information regarding the tubular structure as compared with the structural-steel structure?

R. B. WOODWORTH.—The tube, as you know, is the ideal compression member. The pipe derrick is economical as compared with the structural-steel derrick because it makes use of that fundamental structural advantage; *viz.*, the use of a lightweight material with a large radius of gyration. This advantage is somewhat offset by the fact that new pipe is, as a rule, a much more expensive material than structural steel. The pipe derrick as manufactured at present is a perfect structure, but beyond the derrick, pipe has no structural fitness. The crown blocks, ladders, foundation, machinery supports, and other similar parts of the rig must necessarily be made of structural steel.

E. G. SPILSBURY, New York, N. Y.—With reference to the details that Mr. Woodworth has just given about the effect of lightning on

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a steel structure, he calls attention to the fact that the head blocks were wood and that they were burned out. My attention was called a little time ago to some details about the lightning effect on steel tanks in the Louisiana field. I was told of a very curious phenomenon. One of the managers had made a new departure in that he had erected his tanks without any wooden roofs and indeed without any wooden covers to them. During all his administration, some four or five years, the company never lost a tank. When he was removed to another field, the new superintendent, always having been accustomed to wooden roofs, was very much exercised by the fact that that protection did not exist here. Having occasion to erect a new tank, he constructed it in the ordinary way, with a wooden roof. Inside of three months that tank was struck by lightning and the whole of his oil was lost and burned up. Being rather a pigheaded sort of fellow, he made up his mind that this was an exceptional case, and could not be considered as really demonstrating a fact. So he rebuilt that tank, and rebuilt it with a wooden cover. I understand that in less than three months this second wooden-roofed tank was struck, and that time he lost another $50,000 worth of gasolene. The real explanation of it was, I think, that the electric current striking the tank was not dissipated at once, as in the case of open-steel surfaces, but ran down to a point where the wooden roof and the steel tank were separated by only a few inches, jumped across, and at once perforated the steel tank, and, therefore, it looks to me as though possibly there is danger in having any composite material in the construction of the derrick.

Mr. Woodworth says that the wood on the top of his steel derrick was burned, but the derrick was not injured. In this case the wooden roof was struck, and at the point where the current jumped off onto the side of the tank, the tank was perforated and the oil burned.

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The Control of Petroleum and Natural Gas Wells

BY ALFRED G. HEGGEM,* M. E., TULSA, OKLA.

(New York Meeting, February, 1916)

IT is the purpose of this article to describe methods recently introduced into the oil and natural gas industry to safeguard the lives of the workmen and to protect property from destruction. Only such drilling methods are described as pertain directly to the operations necessary to maintain the well under complete control at all times.

Wells Formerly Drilled Without Controlling Equipment

It may seem incredible that practically all wells, until recently, were drilled without the provision of any means for control. To drill a hole into a boiler carrying high-pressure steam without first providing means to control the escape of steam would be unthinkable, yet in oil fields wells are drilled into a gas or oil formation with a possible pressure of 300 to 1,800 lb. per square inch and a daily flow of 20,000,000 to 80,000,000 cu. ft. of gas or 1,000 to 15,000 bbl. of oil per day. Some wells have produced in excess of 100,000 bbl. of oil per day, but these are exceptional cases. In addition to the loss of product, one must consider the great danger attendant upon the promiscuous liberation of such an amount of explosive and inflammable substance.

Introduction of Metal Casing for Deep Drilling

The introduction of metal casing to secure a permanent and impervious wall in a well made the drilling of deep wells possible and removed many of the original problems encountered in well drilling, but it did not insure the control of the well. To obtain the full value of the casing it was essential that a pressure-tight joint be made between the wall of the well and the lower end of the casing. Many materials in various forms were used, including cotton, hemp, stable refuse, oats, rice, cloth, leather, cork, rubber, lead, clay and cement, resulting, after a slow process of evolution, in the general adoption of two types of bottom fittings for

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casing. These are the "long shoe," having a diameter approximately that of the casing couplings, and fitting into a hole especially prepared to insure a solid and close contact; and the rubber "packer," consisting of a rubber cylinder forced into tight contact with the wall of the well and the exterior of the casing respectively.

The other materials mentioned are used now only under special conditions, except clay and cement, each of which is used extensively, generally in combination with either the long shoe or with a packer.

Under general conditions, the weight of the casing, together with the friction between the casing and the walls of the well, is sufficient to withstand the lifting force of the gas or oil. The friction between the casing and the walls of the well is likely to vary greatly, and is uncertain at best. Additional friction is often obtained by cementing the casing within the well, and where very high pressures are encountered it is customary to secure together the several strings of casing by means of clamps, thus adding the weight of the outer strings of casing, as well as the friction, to that of the inner string.

Another practice is to bury heavy wood sills some distance below the derrick floor and, by means of heavy anchor bolts and suitable clamps, make connection to the top of the casing.

Gate Valve Does not Insure Control

With the bottom of the casing securely in place, and forming a pressure-tight, non-leaking joint, the top of the casing remains to be considered in the matter of control. Valves of various types were tried, and while not accomplishing as much as was desired, they were in many cases a necessity.

In its highest form, this type of equipment consisted of a gate valve, of inside diameter greater than that of the casing, surmounted by the casing head, the two being connected by a short nipple. This arrangement provided the usual casing head to which flow connections could be made and through which the usual drilling operations could be conducted. In addition, the gate valve furnished means by which the well could be shut in when the drilling tools were removed.

It was thought by many that a gate valve so placed on the head of a well insured control, but in practice many failures of this arrangement demonstrated that it did not safeguard either life or property and really caused a false feeling of safety. In fact, because of inability to close the valve promptly at a critical moment, oil or gas has become ignited and burned with great violence, making approach to the well impossible. The gate valve had to be removed by shooting off with a cannon ball, or other means, before the well, thus supposedly safeguarded, could be brought under control.

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Requirements for Efficient Controlling Device

A proper closing and controlling device for the head of an oil or gas well must meet the following requirements: It must permit all drilling operations to be carried on without interference; permit of immediate and tight closing; control the flow without back pressure; insure safety to workmen; be simple in construction; compact in size; sufficiently strong to control maximum pressures; proof against injury in handling; unaffected by sand; and unaffected by fire.

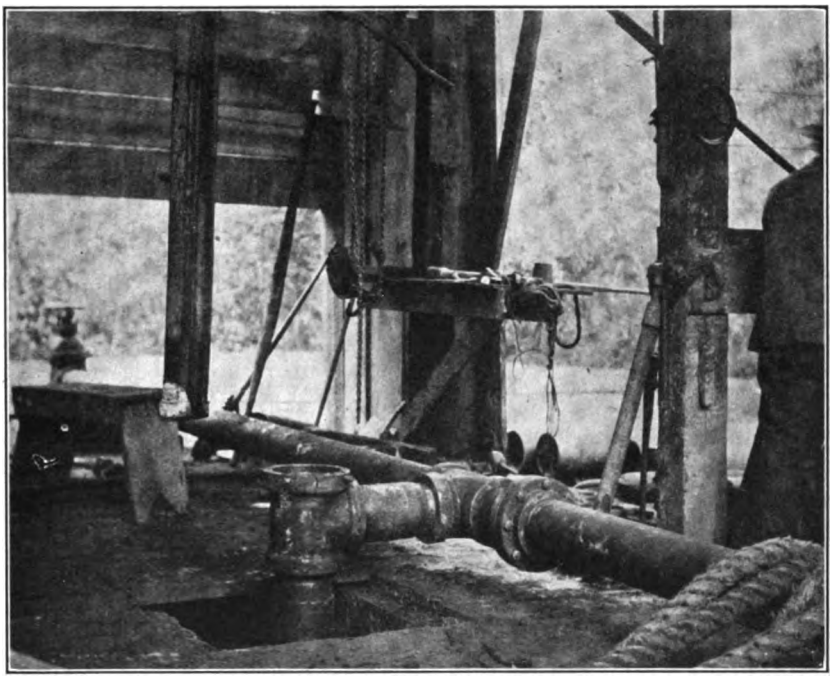

FIG. 1.—OIL WELL FITTED WITH FLOW LINES BEFORE "DRILLING IN." GATE VALVES ARE PLACED ON EACH BRANCH OF THE FLOW LINE BUT THE WELL ITSELF IS WITHOUT MEANS OF CONTROL. THE COMMON CASING HEAD IS CAPPED WITH A SEPARATE TOP WHEN FLOW PRESSURE SUBSIDES SUFFICIENTLY TO PERMIT.

The Control Casing Head

The "control casing head," combining the functions of a gate valve and a casing head, was designed to meet these requirements, which are considered necessary to safeguard life and property during the operations of well drilling.

This device is similar in general appearance and size to the common type of casing head in general use (Fig. 2). It can be placed above or below the derrick floor, at the will of the operator, and is arranged to

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receive the standard fittings commonly used with casing heads. The top opening is threaded to receive a drilling nipple or other top connections usually employed in gas wells.

FIG. 2.—THE CONTROL CASING HEAD.

The interior of the head is bored out to a true cylindrical form into which is closely fitted the plug or valve (Fig. 3). This valve is open at one end to provide a lateral passage for the oil or gas; the other end is

FIG. 3.—DETAIL OF VALVE PLUG AND BODY.

reduced in diameter to form a stem, which extends through a suitable stuffing box, and by which the valve may be operated. On the stem side of the valve a flat surface, or flange, fits closely against the base of the stuffing box, making a tight joint, thereby to a large degree relieving the

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stuffing box of duty in preventing leakage. The extending stem is hexagonal in form to accommodate a wrench, but a transverse hole through it provides a more convenient means of operating by use of a bar of iron, such as a bolt or piece of 1-in. pipe.

To provide for the convenient operation of the valve at a distance, when the casing head is below the floor or is otherwise not readily accessible, the end of the stem is bored out and threaded to take an extension of standard 2-in. pipe.

The back of the valve is broad enough to close completely either top or bottom opening in the body, and provide sufficient lap to prevent leaking.

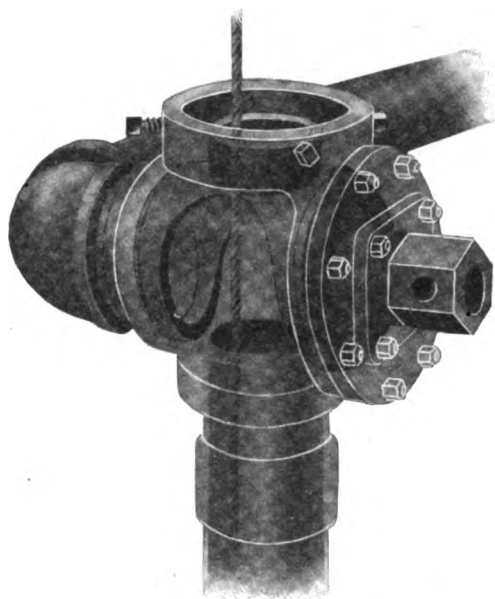

FIG. 4.—PHANTOM VIEW SHOWING TOP OPENING CLOSED TO DEFLECT FLOW INTO TANK. THE DRILLING LINE IS SHOWN IN POSITION IT ASSUMES WHEN VALVE IS CLOSED WITHOUT WITHDRAWING THE DRILLING TOOLS.

On each side of the back of the valve is a groove, or notch, of sufficient size to encompass the drilling line, sand line, or torpedo line. By this provision the valve, when closed, while completely shutting in any flow, does not injure the line.

By means of the end opening, as well as by recessing the back of the valve, the pressures within the casing head are to a large degree counterbalanced, making the operation of the valve easy.

The device is simple, consisting of but four pieces, and having but one moving piece. It cannot be damaged by any of the usual drilling operations or by rough handling.

Stops are provided within the body to prevent the valve from being

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turned too far in either direction. With the valve in normal position but a quarter turn of the stem is required to close the top or the bottom opening, turning the flow into the tanks or entirely shutting in a well.

The control casing head is used in much the same manner as gate valves and common casing heads, but it has many special uses that have developed largely from experience with the great number in operation in various oil fields.

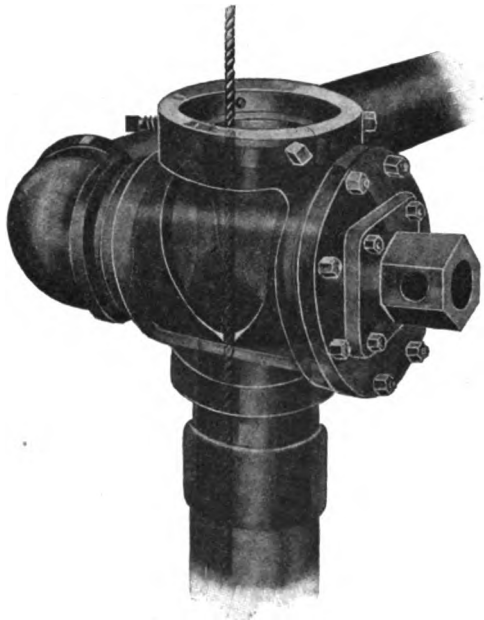

FIG. 5.—PHANTOM VIEW SHOWING WELL SHUT IN ENTIRELY.

The Control of Wildcat Wells

When drilling a well in unknown territory it is not possible to foresee what conditions may be encountered, and many wells so drilled have "gone wild" with great destruction of property and in a number of cases with loss of life.

Some such wells have ruined fields of good prospective value by letting water in on the oil in such quantities as to render further operations in that district unprofitable.

For such cases it is desirable to place a control casing head on the first string of casing landed in the well and maintain it there through all subsequent drilling operations. This insures that the well can be kept under control, and if at any time oil or gas is encountered unexpectedly the well can be instantly shut in until provision is made for taking care of the yield. It is not necessary to withdraw the tools as the valve will close tightly around the line, so no time need be lost in bringing the well

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under control. Also, with the flow stopped or controlled, there is no force acting to expel the tools, so this danger is avoided.

When a new string of casing is set inside of the one carrying the controlling device, the string may be set with the top below the head in order not to interfere with the free action of the valve; or the control head may be set on the inner string by using a swage nipple; or a smaller head of suitable size may be used.

If it is desired to shut in gas between the two strings of casing, a packing ring may be screwed into the larger casing head and a gas-tight joint made on the outside of the inner casing.

This process may be repeated with each string of casing inserted in the well. Of course where several heads are used on the same well, the larger ones should be placed below the derrick floor in order that the last casing head may not extend to such a height above the floor as to interfere with the ready wrenching of the tools or limit the length of temper screw that may be let out.

Use of Control Head in Established Oil Field

When drilling in an established field the control casing head is placed on the last string of casing to be seated in the well. Since it is about the same height as the common casing head, it does not interfere with the wrenching of the tools or other drilling operations.

If a pocket of gas is encountered, the well may be instantly shut in without removing the tools, all fires may be safely extinguished and the boiler moved back to a safe distance, when drilling may be resumed.

On reaching the oil sand, if the well starts to flow it may be shut in, or if flow-line connections have been made the entire flow may be diverted to the flow tank. When the flow ceases, drilling may be resumed and continued without loss of time until the well again starts to flow.

If the well flows continuously, an oil saver is placed on the line immediately above the control casing head, while same is closed. When ready, the valve is turned to allow the oil saver to enter the seat in the top of the casing head. This may be accomplished without loss of oil, wells producing as high as 10,000 and 12,000 bbl. of oil per day having been drilled in this manner without any oil showing on the derrick.

The prevention of waste of oil in drilling means not only a saving to the operator, but also the reduction of the fire hazard to a minimum. The driller and "tool dresser" do not get drenched in oil and thereby risk fatal injury by fire, which under the usual conditions prevailing before the introduction of the control casing head occurred with appalling frequency.

Moving the boiler away from the rig did not remove all hazard, for cases are known in which a tool dresser, in clothes soaked with oil from a spraying well, has approached the boiler to try the water cocks

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and had the flames from under the boiler set fire to his clothing. Such accidents are almost always fatal.

The Control Head Useful in Shooting an Oil Well

It often happens that, while a well still flows or sprays oil continuously, the yield has reduced to such an extent that shooting must be resorted to in order to save the well. There are, in fact, many reasons for shooting a well, even though the flow pressure is such as to make the placing of the shot a hazardous undertaking.

In the history of the petroleum industry, one finds that frequently a well would flow during the placing of a shot, causing the shell to be ejected from the well to explode in the derrick, with frightful loss of life and property.

The use of the control casing head on the inner string of casing makes the shooting of any well a safe operation. The shell containing the "shot" is lowered until the top is below the casing head, when the valve in the latter is turned to close the top opening, and the oil, which under former conditions would spray into the air and be lost, is now all caught in the flow tank. The slight opening in the rope notch of the valve to permit the torpedo line to slip through is not sufficient to permit an appreciable amount of waste.

The valve is maintained in this closed position until the shell is landed on bottom and the torpedo line withdrawn. It is then opened to permit the introduction of the squib, or other means employed to explode the shot, and is kept open until the well has cleaned itself, when it may be closed in time to catch the "second flow."

While usual practice demands that the well be open during shooting to permit the well to cleanse itself of the torpedo shells and of the sand and other material loosened by the shot, yet by the use of the control casing head all of this may be deflected into a tank without putting a back pressure on the well. Not only will all of the oil be saved, but a more complete record of the effect of the shot will be found in the accumulations arising from fragments of rock broken off and thus trapped.

The introduction of the control casing head has made it possible to shoot wells of any size regardless of the amount of flow, and it has in practice demonstrated its value by checking the flow in a well which threatened to expel the shot, consisting of 20 qt. of nitroglycerine contained in two shells joined together. In this case, of course, the valve was turned to close the bottom opening and entirely shut in the well. This was done instantly and without injury to the torpedo line.

Application of Control Head to Gas Wells

While the name "casing head" would seem to imply a use limited to oil wells, the control casing head is equally useful in drilling gas wells, and is used in the same manner as in drilling wells for oil.

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In many places it is customary to drill gas wells during daylight only and to allow the well to remain open during the night, thus wasting a large volume of gas and reducing the rock pressure on the gas that remains in the sand after the well is completed. With the control casing head on the well the gas can be shut in during the time when active drilling operations are suspended. Not only is the waste of gas stopped and a higher rock pressure insured, but also the fire hazard is practically eliminated.

When the well is completed the control casing head is utilized in the same manner as the common T fitting. The usual top and side connections are made to the head, with the advantage that in the event of leaky gate valves, or damage to the lines, the gas may be instantly shut in and repairs made without waste of gas or danger of fire.

After a gas well is completed, it is the usual custom to limit the flow of gas into a pipe line by partially closing the valve at the head of the well. This, in the case of a gate valve, leaves a crescent-shaped opening of great length and slight width, with the result that all of the gas in passing through is forced to rub the sides of the opening, causing rapid wear of the gate and seats and destruction of the valve. The control casing head, in contrast, gives a small round opening gradually increasing to an oval-shaped opening, thus presenting a minimum rubbing surface in contact with the gas and reducing the wearing action.

The tight closing of the valve in the control casing head and the freedom from leakage are very important advantages in securing long life and freedom from accidents.

Effect of Sand on Control Casing Head

Owing to the absence of recesses and projections within the body of the control casing head, there is no place in which sand can lodge to interfere with the action of the valve. The sand carried by flowing oil during drilling operations, therefore, has not in any way disturbed the easy operation of the valve.

While sand cannot lodge and interfere with the action of the valve, there is a possibility of injury from the cutting action of sand. To meet this condition the valve portion is made with a curved surface to deflect the flow gradually from vertical, in the casing, to horizontal, in the flow line. This curved surface is backed by a great thickness of tough metal which will for a long time resist the scouring action of the sand. By thus gradually changing the direction of flow, no flat surface is exposed to the direct impact of the sand and cutting action is materially reduced.

Some operators prefer to cushion the flow when the well is making much sand. This can readily be done by screwing a short joint of pipe

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into the top of the casing head and closing the top of the pipe by a metal plug. Sometimes a plug of wood is driven into the cushion pipe to protect the metal plug from being cut by the sand.

This cushion pipe may be assembled and secured in place, while the casing head is closed, after which the valve may be turned to the side position out of the path of the oil and sand.

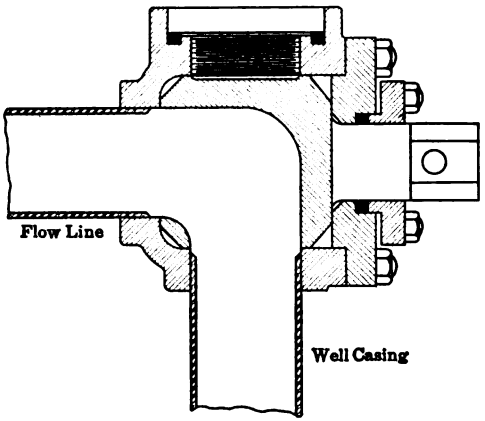

FIG. 6.—CONTROL CASING HEAD ON WELL SHOWING THE CURVED DEFLECTING SURFACE TO CHANGE GRADUALLY THE DIRECTION OF FLOW AND PREVENT THE MECHANICAL VAPORIZATION OF THE OIL. THE AREA OF THE PASSAGE THROUGH THE CASING HEAD BEING GREATER THAN THAT OF THE CASING INSURES A REDUCED VELOCITY AND PREVENTS FRICTIONAL BACK PRESSURE ON THE WELL.

The Control Head as an Aid in Killing a Gas Well

Gas is frequently encountered in drilling for oil and interferes with the further drilling of the well. Experience has shown that gas may be found in large volume and at high pressure above a formation containing oil in paying quantities. This fact has led to enormous waste of gas in an endeavor to reduce the pressure so that drilling could be continued to the deeper oil-bearing formation.

For a long time this waste of gas was considered necessary and was condoned on the ground that the value of the gas so wasted was less than that of the oil subsequently secured.

Aside from the loss due to the waste of the gas, the delay in drilling adds greatly to the expense of the well, and postpones the recovery of oil, entailing a further loss of oil through offset wells which have previously tapped the oil sand and are drawing from the common pool.

It behooves the operator to complete his well in the shortest time possible; therefore, the modern method is to seal off the gas sand and continue drilling without further delay.

The use of mud-laden fluid in killing a gas well is too well known to require a description here, but it may be interesting to know that the

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"lubricator" method of introducing mud into a gas well presented some practical difficulties in operation and was attended with some hazard.

The large gate valves which were used became difficult to operate, partly because they were not designed to operate in mud.

By using the control casing head already on the well, these difficulties and the expense of buying and the loss of time in securing special gate valves are avoided.

On top of the control casing head are set one or more joints of any size casing available (two joints of 10-in. casing are my own preference). To

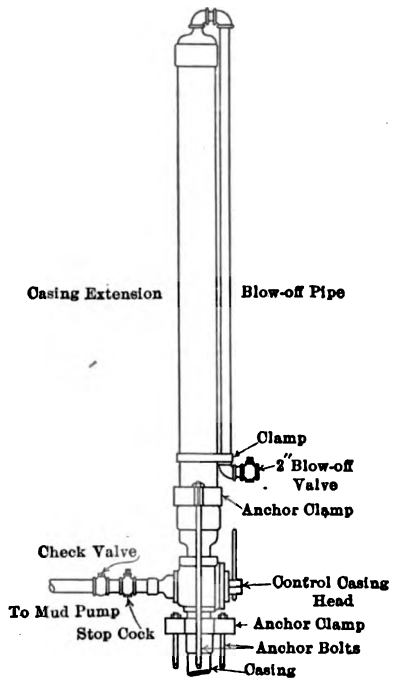

FIG. 7.—"LUBRICATOR" OF IMPROVED FORM FOR INTRODUCING MUD FLUID INTO A WELL TO "KILL GAS." ALL OPERATIONS ARE CONTROLLED FROM THE DERRICK FLOOR WITHOUT EXTRA HELP BEING REQUIRED.

the top of the casing extension thus provided, a 2-in. pipe line is connected and brought down the outside of the extension to within about 4 ft. of the floor, the end being fitted with a valve or stop cock (Fig. 7).

From the side outlet of the control casing head a 2- or 3-in. connection is made to the pump discharge. This line should be provided with a check valve and a stop cock.

All fittings should be of suitable strength, and as the control casing head is designed for a safe load of 1,800 lb. per square inch all other fittings should be selected from extra heavy stock.

The 2-in. down pipe from the top of the extension should be securely

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clamped to the extension, and the valve-controlled outlet should be turned in a direction away from the operator.

These fittings are all made while the well is shut in. The pump is then started and mud-laden fluid pumped into the extension chamber until it is full, as indicated by mud showing at the outlet of the blow-off pipe. This outlet is then closed and the valve in the casing head opened permitting the mud to pass into the well. As soon as the extension chamber has emptied, the valve in the casing head is again closed and the outlet valve on the down pipe opened. This operation is repeated until the well is filled and the gas killed.

The advantage of this arrangement lies in the fact that everything is controlled from the derrick floor and no extra help is required. Also time is saved, since the pump will start delivering mud into the extension chamber as soon as the gas pressure is reduced, and, further, it is possible to pump directly into the well without any change of fittings.

Control Head on Flowing Well Reduces Oil Loss and Fire Hazard

With the old method of closing in a flowing well, a flat top was set in the old-style casing head several inches above the side outlet. The column of oil would strike this flat top at high velocity and be broken up into a fine spray, which, upon mixing with the gas, would appear as a blue "smoke" over the flow tank. This spray would float in the air for a considerable distance, wasting a large amount of oil and covering the surrounding trees and grasses with an inflammable oil. It would also settle in so-called "gas pockets" along the road, frequently becoming ignited by passing automobiles, with attendant loss of life and property.

As previously mentioned, the curved form of the valve in the control casing head gradually deflects the current of oil into the flow line. Therefore, the oil is not broken up into a spray, more good oil is put into the tank, and the fire hazard is reduced.

The Control Casing Head Now Largely Used

The control casing head fills a want that has been long felt and is rapidly being adopted as standard equipment in the drilling of wells for petroleum and natural gas.

At present (September, 1915) approximately 1,000 control casing heads are in use, and, although a great variety of conditions has been encountered in drilling wells, ranging from dry holes to oil wells producing 12,000 bbl. a day, and gas wells having a rock pressure of 1,150 lb. per square inch and volumes in excess of 40,000,000 cu. ft. per day, there has not been a single failure of the head to safeguard life and property, and not one well has burned or become unmanageable.

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Development of the Law Relating to the Use of Gas Compressors in Natural Gas Production†

BY SAMUEL S. WYER,* M. E., COLUMBUS, OHIO

(New York Meeting, February, 1916)

THE art of natural-gas compressing is now over 25 years old, and has grown at practically the same rate as the increase in domestic natural-gas consumers. There are now over 200 natural-gas compressing stations in North America, aggregating more than 320,000 hp. of compressor capacity and representing a property value of more than $22,000,000, and compressing more than 85 per cent. of all the gas used. The age and magnitude of the art make it evident that the use of gas compressors is a recognized integral part and universal custom of the natural-gas business.

The public also is not without its rights and vital interests in this problem. Approximately 1,700,000 domestic natural-gas consumers in North America are dependent upon gas compressors for their natural-gas service. That is, if the use of compressors were to be prohibited, the majority of these consumers would be unable to secure adequate natural-gas service.

Each consumer represents between four and five persons, and it therefore follows that the comfort and well-being—as far as natural-gas service is concerned—of at least 8,000,000 persons would be affected if the use of natural-gas compressors should be prohibited. These 1,700,000 consumers have invested, in services, house piping, fixtures, and appliances, an average of about $90 each, or an aggregate of $153,000,000, which is much more than the companies' investment in gas compressors. Furthermore, in all cases where the rate paid by the consumers is fair to the gas company—considering the value of the service rendered by the gas company—the consumers are entitled to continued service and protection of the investment they have made on the faith that the gas service would be continued in the future.

Since "Customs adopted and acquiesced in, if not in conflict with federal or State legislation, have the force of positive law,"[1] and "Courts will take notice of whatever ought to be generally known within the limits of their jurisdiction,"[2] there ought to be no question as to the un-

* Consulting Engineer.

† Reference should be made to the succeeding paper, Necessary Use and Effect of Gas Compressors on Natural Gas Field Operating Conditions; see also Is It Feasible to Make Common Carriers of Natural Gas Transmission Lines?, vol. xlviii, 471 to 480 (1914).

[1] Lindley on Mines, Sec. 271, vol. i, 3rd ed.

[2] U. S. Supreme Court, Brown vs. Spilman, 155 U. S., 670.

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qualified right to use gas compressors. However, many small gas-producers, not using gas compressors, have sought to secure permanent injunctions from courts to prevent other gas-producers from using such compressors. This has resulted in much expensive litigation. The following is a chronological arrangement of all the court decisions relating to this question, and shows the logical development of the common law. To expedite cross-reference work, each case is given a serial number; also, abbreviations used in citations are given in full in the accompanying table.

Ch. Div.....Chancery Division, British Law Reports.
N. E.....Northeastern Reporter.
Ind.....Indiana Reports.
Fed. Rep.....Federal Reporter.
U. S.....United States Reports.
Pa.....Pennsylvania State Reports.
Pac. Rep.....Pacific Reporter.
S. W. Rep.....Southwestern Reporter.
Ohio State.....Ohio State Reports.
Oklahoma.....Oklahoma Reports.

There are four classes of cases, namely, those that relate to:

Common-law rights in use of gas compressors. Nos. 1, 3, 7, 10, 12, 13, 26.

Rights under State laws regulating use of gas compressors. Nos. 4, 14, 16, 17, 19, 20.

Judicial recognition of declining gas volumes. Nos. 7, 8, 10, 12, 13, 21, 22, 23, 29,.

Questions of "ownership," "possession," or "right of transportation" of natural gas. Nos. 2, 5, 6, 7, 8, 9, 11, 13, 15, 18, 21, 24, 25, 27, 28, 29, 30, 31.

1885

No. 1.—Ballard vs. Tomlinson, 29 Ch. Div. 122.

"The plaintiff, if he has a right to use anything in nature, has the right to exercise that use by all the skill and invention of which man is capable, and it seems to me that as long as the plaintiff uses only lawful means as against his neighbor, however ingenious and however artificial those means may be, his right to appropriate the common source is not diminished because he uses the most artificial or the most ingenious methods." Cited in:

No. 12.—Jones vs. Forest Oil Co., 194 Pa. 379.

1889

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., Supreme Court of Indiana, 22 N. E. 778.

"Natural gas is as much an article of commerce as iron ore, coal, petroleum, or any other of the like products of the earth. It is a commodity which may be transported, and it is an article which may be bought and sold in the markets of the country. * * * The power to regulate commerce between the States is exclusively in the federal congress. An action by congress will not authorize the States to legislate in matters of interstate commerce." * * * Transportation of commercial commodities

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from State to State is interstate commerce, and the State legislatures can neither burden nor restrict it." Cited in:

No. 4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.

5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

9.—Townsend vs. State, 147 Ind. 624.

15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.

18.—Federal Oil Co. vs. Western Oil Co., 121 Fed. Rep. 674.

19.—Richmond Natural Gas Co. vs. Enterprise Nat. Gas Co., Appelate Court of Indiana, 66 N. E. 782.

28.—Haskell vs. Cowham, 187 Fed. Rep. 403.

30.—West vs. Kansas Nat. Gas Co., 221 U. S. 229.

1889

No. 3.—Westmoreland, etc., Co. vs. DeWitt, Supreme Court of Pennsylvania, 130 Pa. 235.

"Gas, it is true, is a mineral; but it is a mineral with peculiar attributes, which require the application of precedents arising out of ordinary mineral rights, with much more careful consideration of the principles involved than of mere decisions. Water also is a mineral; but the decisions in ordinary cases of mining rights, etc., have never been held as unqualified precedents in regard to flowing, or even to percolating, waters. Water and oil, and still more strongly gas, may be classed by themselves, if the analogy be not too fanciful, as minerals *ferœ naturæ*. In common with animals, and unlike other minerals, they have the power and the tendency to escape without the volition of the owner. Their 'fugitive and wandering existence within the limits of a particular tract was uncertain' * * * They belong to the owner of the land, and are a part of it, so long as they are on or in it, and are subject to his control; but when they escape, and go into other land, or come under another's control, the title of the former owner is gone. Possession of the land, therefore, is not necessarily possession of the gas. If an adjoining, or even a distant, owner drills his own land, and taps your gas, so that it comes into his well and under his control, it is no longer yours, but his * * * the one who controls the gas—has it in his grasp, so to speak—is the one who has possession in the legal as well as in the ordinary sense of the word." Cited in:

No. 5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

8.—Brown vs. Spilman, 155 U. S. 665.

9.—Townsend vs. State, 147 Ind. 624.

12.—Jones vs. Forest Oil Co., 194 Pa. 379.

13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

25.—Kansas Natural Gas Co. vs. Haskell, 172 Fed. Rep. 545.

1891

No. 4.—Jamieson vs. Indiana Natural Gas & Oil Co., Supreme Court of Indiana, 128 Ind. 555.

"The Indiana statute prohibiting the transportation of natural gas through pipes at a greater pressure than 300 lb. per square inch, or otherwise than by its natural flow, is a valid exercise of the police power of the State, since natural gas is an intrinsically dangerous substance, and the legislature having determined what pressure is reasonable and safe the courts cannot review its action. * * * That natural gas is a dangerous agency is a matter of common knowledge; and hence courts take judicial notice of that fact. We know, as the legislature knew, and as every one knows, that natural gas is in a high degree inflammable and explosive. * * * It would be unreasonable to hold that the courts know judicially that natural gas is a public necessity so far as to warrant the exercise of the right of eminent domain, and yet hold that they do not know that it is inflammable and explosive. Knowing the one thing, they must

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know the other. We hold without hesitation that natural gas is so dangerous that its use may be made the subject of a police regulation. Decision after decision recognizes the principle we have stated, and upholds laws regulating the use of property. * * * The public safety and welfare is the highest consideration in all legislation, and to this consideration private rights must yield. No man has a right to so use a dangerous species of property as to put the safety of others in peril. Liberty does not imply the right of one man to so use property as to endanger the property of others; nor does ownership imply any such right. This is rudimental. It must therefore be true that the owner of property of such a dangerous nature as to require regulation to prevent injury to others can have no right paramount to the police power. It is not too much to say that, as against the police power, there is no such thing as a vested right. * * * No investment, however great, can so vest a right as to preclude the just exercise of a great governmental power, such as that under which regulations for the protection of the health and safety of persons are enacted. This principle is supported by many decisions. * * * We have already declared that it is a dangerous substance, requiring regulation, and we shall only add to what we have said a quotation from the opinion. * * * 'It was not necessary,' said the court, 'to aver that coal-oil is inflammable, or to prove it. Courts and juries will take cognizance of such matters as are of common knowledge, and pertain to the experience and affairs of almost every man's daily life. Courts do not require proof that fire will burn, or powder explode, or gas illuminate, or that many other processes in nature and art produce certain known effects.'" Cited in:

No. 5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

9.—Townsend vs. State, 147 Ind. 624.

15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.

19.—Richmond Natural Gas Co. vs. Enterprise Natural Gas Co., 66 N. E. 782.

(It is important to note that Case No. 19 distinguishes between the use of gas compressors for transporting gas and for pumping wells.)

25.—Kansas Natural Gas Co. vs. Haskell, 172 Fed. Rep. 545.

30.—West vs. Kansas Natural Gas Co., 221 U. S. 229.

1892

No. 5.—People's Gas Co. vs. Tyner, Supreme Court of Indiana, 131 Ind. 277. "It has been settled in this State that natural gas, when brought to the surface of the earth and placed in pipes for transportation, is property, and may be the subject of interstate commerce. Water, petroleum oil, and gas are generally classed by themselves as minerals possessing in some degree a kindred nature. As to whether the owner of the soil may dig down and divert a well-defined subterranean stream of water, there is much diversity of opinion and conflict in the adjudicated cases; but the authorities agree that the owner of a particular tract of land may sink a well and appropriate to his own use all the percolating water found therein, though it may entirely destroy the well on his neighbor's land. * * * It is a familiar maxim that in contemplation of law, land always extends downward as well as upward, so that whatever is in a direct line between the surface of any land and the center of the earth belongs to the owner of the surface. * * * When it is once conceded that the owner of the surface has the right to sink a well and draw gas from the lands of an adjoining owner, no valid reason can be given why he may not enlarge his well by the explosion of nitroglycerine therein for the purpose of increasing the flow. The question is not as to the quantity of gas he may take, but it is a question of his right to take the gas at all." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., Supreme Court of Indiana, 22 N. E. 778.

3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.

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Cited in:

No. 6.—Tyner vs. People's Gas Co., 131 Ind. 408.

9.—Townsend vs. State, 147 Ind. 624.

11.—State of Indiana vs. Ohio Oil Co., 150 Ind. 21.

18.—Federal Oil Co. vs. Western Oil Co., U. S. Circuit Court of Appeals, 121 Fed. Rep. 674.

1892

No. 6.—Tyner vs. People's Gas Co. Supreme Court of Indiana, 131 Ind. 408. This merely reaffirms the doctrine in the preceding case. Cites:

No. 5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

1893

No. 7.—Hague vs. Wheeler, Supreme Court of Pennsylvania, 157 Pa. 324.

"A court of equity will not interfere by injunction to compel a landowner who has sunk a gas well on his own premises without malice or negligence to stop the flow of gas therefrom, which has proven insufficient in quantity to enable him to utilize it, at the suit of the adjoining owners, whose wells yield gas in sufficient quantities to enable them to utilize and market it, though defendant's well drains the common reservoir, and thus will ultimately reduce the flow of plaintiffs' wells. * * * The mere fact that the defendants, by operations upon their land, are taking gas from the earth, and thereby diminishing the quantity of gas which would otherwise come to the plaintiffs' wells, furnishes no ground for complaint or equitable interference." In speaking of oil and gas the Court said, "But they are not, like coal and iron ore, fixed in their place in the rocks, so that the owner may know his own, protract his lines downward to mark his boundaries, and take them when he pleases. As water percolates by untraceable rills through the gravel, so these 'minerals *feræ naturæ*,' as they have been aptly called in a recent case, permeate the porous rocks deep in the bowels of the earth, and rush to the surface through any opening made through the impervious cap by which the basin which contains them is sealed. No landowner gets through his wells oil or gas exclusively from his own land. That which saturates his rocks may be lawfully taken by his neighbor through wells on his land, tapping the common reservoir. From the very nature of the case, the right of each owner is qualified. It is common to all whose land overlies the basin, and each must of necessity exercise his right with some regard to the rights of the others. * * * The common law is a growing tree; its principles must be continually adapted to new facts, and the changing conditions of modern life. Only the legislature can grub it up, but the courts are charged with the duty of pruning its branches, and sometimes grafting a new scion on the old stock. * * * it is said that the oil and gas are unlike the solid minerals, since they move through the interstitial spaces or crevices in the sand rocks in search of an opening through which they may escape from the pressure to which they are subject. This is probably true. It is one of the contingencies to which this species of property is subject. But the owner of a surface is an owner downward to the center, until the underlying strata have been severed from the surface by sale. What is found within the boundaries of his tract belongs to him according to its nature. The air and water he may use. The coal and iron or other solid minerals he may mine and carry away. The oil and gas he may bring to the surface and sell in like manner, to be carried away and consumed. His dominion is, upon general principles, as absolute over the fluid as the solid minerals. * * * He cannot estimate the quantity in place of gas or oil, as he might of the solid minerals. He cannot prevent its movement away from him, toward an outlet on some other person's land, which may be more or less rapid, depending on the dip of the rock or the coarseness of the sand composing it; but so long as he can reach it and bring it to the surface it is his absolutely, to sell, to use, to

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give away, or to squander, as in the case of his other property. In the disposition he may make of it he is subject to two limitations: he must not disregard his obligations to the public; he must not disregard his neighbor's rights. If he uses his product in such a manner as to violate any rule of public policy or any positive provision of the written law, he brings himself within the reach of the courts. If the use he makes of his own, or its waste, is injurious to the property or the health of others, such use or waste may be restrained, or damages recovered therefor; but, subject to these limitations, his power as an owner is absolute, until the legislature shall, in the interest of the public as consumers, restrict and regulate it by statute." Cited in:

No. 13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

1895

No. 8.—Brown vs. Spilman, U. S. Supreme Court, 155 U. S. 665.

"Petroleum gas and oil are substances of a peculiar character, and decisions in ordinary cases of mining for coal and other minerals which have a fixed situs, cannot be applied to contracts concerning them without some qualifications. They belong to the owner of the land, and are part of it, so long as they are on it or in it or subject to his control; but when they escape and go into other land, or come under another's control, the title of the former owner is gone. If an adjoining owner drills his own land, and taps a deposit of oil or gas, extending under his neighbor's field, so that it comes into his well, it becomes his property. * * * When oil or gas is found in paying quantities, it is not usual to consume it or reduce it to use at the wells, but it is conducted in iron pipes to large tanks or reservoirs, whence it is distributed by other pipes to the places of consumption, often many miles distant. These are matters within the common experience or knowledge of all men living in those portions of the country where oil and gas are produced, and courts will take notice of whatever ought to be generally known within the limits of their jurisdiction." Cites:

No. 3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

Cited in:

No. 13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

22.—Brewster vs. Lanyon Zinc Co., U. S. Circuit Court of Appeals, 140 Fed. Rep. 801.

25.—Kansas Natural Gas Co. vs. Haskell, U. S. Circuit Court, 172 Fed. Rep. 545.

1897

No. 9.—Townsend vs. State, Supreme Court of Indiana, 147 Ind. 624.

Affirms State's right in preventing the burning of natural gas in flambeau lights on account of the wasteful use of natural gas in this manner, since the wasteful use of gas which is drawn from a common storage not reduced to individual possession is an injury to others. Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N. E. 778.

3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.

5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

Cited in:

No. 11.—State of Indiana vs. Ohio Oil Co., 150 Ind. 21.

1897

No. 10.—Thomas C. Kelly vs. Ohio Oil Co., Ohio Supreme Court, 57 Ohio State 317.

"When a person has the legal right to do a certain act, the motive with which it is done is immaterial. The right to acquire, enjoy, and own property carries with it the right to use it as the owner pleases, so long as such use does not interfere with the legal

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rights of others. * * * Whatever gets into the well belongs to the owner of the well, no matter where it came from. In such cases the well and its contents belong to the owner or lessee of the land, and no one can tell to a certainty from whence the oil, gas, or water which enters the well came, and no legal right as to the same can be established or enforced by an adjoining landowner. The right to drill and produce oil on one's own land is absolute, and cannot be supervised or controlled by a court or an adjoining landowner. So long as the operations are legal, their reasonableness cannot be drawn in question. * * * Petroleum oil is a mineral, and while in the earth it is part of the reality, and, should it move from place to place by percolation or otherwise, it forms part of that tract of land in which it tarries for the time being, and, if it moves to the next adjoining tract, it becomes part and parcel of that tract; and it forms part of some tract until it reaches a well, and is raised to the surface, and then for the first time it becomes the subject of distinct ownership, separate from the realty, and becomes personal property—the property of the person into whose well it came. And this is so whether the oil moves, percolates, or exists in pools or deposits. In either event, it is the property of, and belongs to, the person who reaches it by means of a well, and severs it from the realty, and converts in into personalty. While it is generally supposed that oil is drained into wells for a distance of several hundred feet, the matter is somewhat uncertain, and no right of sufficient weight can be founded upon such uncertain supposition to overcome the well-known right which every man has to use his property as he pleases, so long as he does not interfere with the legal rights of others. * * * While the drilled oil well is artificial, the pores and channels through which the oil reached the bottom of the well are natural." Cited in:

No. 25.—Kansas Natural Gas Co. vs. Haskell, 172 Fed. Rep. 545.

1898

No. 11.—State of Indiana vs. Ohio Oil Co., Indiana Supreme Court, 150 Ind. 21.

"The title to natural gas does not vest in any private owner until it is reduced to actual possession. A statute making it unlawful to permit the escape of natural gas into the open air from a well for longer than two days after it is constructed is not unconstitutional. The continuous and persistent waste of natural gas to the detriment of the community at large and in violation of statute is a nuisance which may be abated by injunction. * * * In the light of these facts, one who recklessly, defiantly, persistently, and continuously wastes natural gas, and boldly declares his purpose to continue to do so, as the complaint charges appellee with doing, all of which it admits to be true by its demurrer, ought not to complain of being branded as the enemy of mankind. * * * It is not the use of unlimited quantities of gas that is prohibited, but it is the waste of it that is forbidden. The object and policy of that inhibition is to prevent, if possible, the exhaustion of the store house of nature, wherein is deposited an element that ministers more to the comfort, happiness, and well-being of society than any other of the bounties of the earth. * * * But we cannot have the blessing of natural gas unless the measures for the preservation thereof in this State are enforced against the lawless. We therefore conclude that the facts stated in the complaint make a case of public nuisance which the appellant has a right to have abated by injunction, and that the complaint states facts sufficient to constitute a cause of action."

Affirmed by U. S. Supreme Court in:

No. 13.—Ohio Oil Co. vs Indiana, 177 U. S. 190.

Cites:

No. 5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

No. 9.—Townsend vs. State, 147 Ind. 624.

Cited in:

No. 25.—Kansas Nat. Gas Co. vs. Haskell, 172 Fed. Rep. 545.

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1900

No. 12.—Jones vs. Forest Oil Co., Supreme Court of Pennsylvania, 194 Pa. 379

"A gas pump may lawfully be used to increase the production of an oil well, although the production of wells on adjoining property is thereby diminished. * * * The question here is to what extent an owner of oil wells may use mechanical devices for bringing the oil to the surface. In operating his own wells, may he use appliances which diminish the production of his neighbor's wells? It is not denied that a gas pump will to some extent affect the production of oil wells located in the immediate neighborhood of the well to which the pump is attached, if the sand from which the oil is obtained is of a porous, pebbly nature, as is the case in the McCurdy field. * * * An owner of land may dig a well upon his property, and if, in so doing, he taps the hidden flow of water, which supplies his neighbor's spring, it is a loss to the neighbor for which the law provides no remedy. * * * 'It must be conceded, we think, that every man is entitled to the ordinary and natural use and enjoyment of his property; he may cut down the forest trees, clear and cultivate his land, although in so doing he may dry up the sources of his neighbor's springs, or remove the natural barriers against wind and storm. * * * In sinking his well he may intercept and appropriate the water which supplies his neighbor's well.' In this case the defendant has the exclusive right to bore for oil. * * * The right being a lawful one, the defendant is at liberty to use all lawful means to obtain all the gas and oil contained in, or obtainable through, the land.' * * * And to that end it may resort to the use of all known lawful modern machinery and appliances. The plaintiff's claim is that the use of a gas pump in the production of oil is unlawful, because, as he alleges, by its powerful suction the oil and gas are drawn from his adjoining farm, thereby decreasing his production. Plaintiff assumes that there is a certain fixed amount of oil and gas under his farm, in which he has an absolute property. True, they belong to him while they are a part of his land; but when they migrate to the lands of his neighbor, or become under his control, they belong to the neighbor. * * * The principle of natural user does not apply at all. The plaintiff, if he has a right to use anything in nature, has a right to exercise that use by all the skill and invention of which man is capable, and it seems to me that as long as the plaintiff uses only lawful means as against his neighbor, however ingenious or however artificial those means may be, his right to appropriate the common source is not diminished because he uses the most artificial or most ingenious methods.' If it is lawful to take water from a substrata by the "exercise of all the skill and invention of which man is capable,' we see no reason why it is not lawful to produce oil by those means, especially as the possession of the soil for purposes of tillage gives the owner no actual possession of the oil and gas underlying it. The evidence shows that the gas pump has been in constant use in all fields, except one, to a greater or less extent, since the discovery of oil; that its use has been generally recognized by all operators; and that it is only used on wells in territory which is almost exhausted. * * * In view of the testimony and authorities above cited, we conclude that the use of a gas pump by defendant, under the circumstances of this case, is not an unlawful act that should be restrained by injunction; that the plaintiff is not entitled to the relief prayed for; and that the bill should be dismissed. Cites:

No. 1.—Ballard vs. Tomlinson, 29 Ch. Div. 194.

3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.

Cited in:

No. 13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

1900

No. 13.—Ohio Oil Co. vs. Indiana, Supreme Court of the United States, 177 U. S. 190.

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* * * "oil and gas, like other minerals, are situated beneath the surface of the earth, but except for this one point of similarity, in many other respects they greatly differ. They have no fixed situs under a particular portion of the earth's surface within the area where they obtain. They have the power, as it were, of self-transmission. No one owner of the surface of the earth, within the area beneath which the gas and oil move, can exercise his right to extract from the common reservoir, in which the supply is held, without, to an extent, diminishing the source of supply as to which all other owners of the surface must exercise their rights. * * * Now, it is doubtless true that the public has a sufficient interest in the preservation of oil and gas from waste to justify legislation upon this subject. Something has been done in this direction already by the acts regulating the plugging of abandoned wells. * * * In the disposition he may make of it (private property) he is subject to two limitations. He must not disregard his obligations to the public. He must not disregard his neighbor's rights. If he uses his product in such a manner as to violate any rule of public policy, or any positive provisions of the written law, he brings himself within the reach of the courts. If the use he makes of his own, or its waste, is injurious to the property or the health of others, such use or waste may be restrained, or damages recovered therefor; but, subject to these limitations, his power as an owner is absolute until the legislature shall, in the interest of the public, as consumers, restrict and regulate it by statute. * * * the rule of property in the state of Indiana to be as follows: Although in virtue of his proprietorship the owner of the surface may bore wells for the purpose of extracting natural gas and oil until these substances are actually reduced by him to possession, he has no title whatever to them as owner. That is, he has the exclusive right on his own land to seek to acquire them, but they do not become his property until the effort has resulted in dominion and control by actual possession. It is also clear from the Indiana cases cited that, in the absence of regulation by law, every owner of the surface within a gas field may prosecute his efforts and may reduce to possession all or every part, if possible, of the deposits, without violating the rights of the other surface owners. * * * But whilst there is an analogy between animals *feræ naturæ* and the moving deposits of oil and natural gas, there is not identity between them. Thus, the owner of the land has the exclusive right on his property to reduce the game therefound to possession, just as the owner of the soil has the exclusive right to reduce to possession the deposits of natural gas and oil found beneath the surface of his land. The owner of the soil cannot follow game when it passes from his property; so, also, the owner may not follow the natural gas when it shifts from beneath his own to the property of someone else within the gas field. It being true as to both animals *feræ naturæ* and gas and oil, therefore, that whilst the right to appropriate and become the owner exists, proprietorship does not take being until the particular subjects of the right become property by being reduced to actual possession. The identity, however, is for many reasons wanting. In things *feræ naturæ* all are endowed with the power of seeking to reduce a portion of the public property to the domain of private ownership by reducing them to possession. In the case of natural gas and oil no such right exists in the public. It is vested only in the owners in fee of the surface of the earth within the area of the gas field. * * * as to gas and oil the surface proprietors within the gas field all have the right to reduce to possession the gas and oil beneath. They could not be absolutely deprived of this right which belongs to them without a taking of private property. * * * In view of the fact that regulations of natural deposits of oil and gas and the right of the owner to take them as an incident of title in fee to the surface of the earth, as said by the Supreme Court of Indiana, is ultimately but the regulation of real property, and they must hence be treated as relating to the preservation and protection of rights of an essentially local character. Considering this fact and the peculiar situation of the substances, as well as the character of the rights of the surface owners, we cannot say that the statute amounts to a taking of private

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property, when it is but a regulation by the State of Indiana of a subject which especially comes within its lawful authority." Affirms:

No. 11.—State of Indiana vs. Ohio Oil Co., 150 Ind. 21.

Cites:

No. 3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.
7.—Hague vs. Wheeler, 135 Pa. 324.
8.—Brown vs. Spilman, 185 U. S. 665.
12.—Jones vs. Forest Oil Co., 194 Pa. 379.

Cited in:

No. 18.—Federal Oil Co. vs. Western Oil Co., 121 Fed. Rep. 674.
19.—Richmond Natural Gas Co. vs. Enterprise Nat. Gas Co., Appellate Court of Ind., 66 N. E. 782.
22.—Brewster vs. Lanyon Zinc Co., 140 Fed. Rep. 801.
25.—Kansas Nat. Gas Co. vs. Haskell, 172 Fed. Rep. 545.
28.—Haskell vs. Cowham, 187 Fed. Rep. 403.
30.—West vs. Kansas Natural Gas Co., 221 U. S. 229.

1900

No. 14.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., Supreme Court of Indiana, 155 Ind. 461.

"The question to be determined here is not whether natural gas, when reduced to possession, is property, but as to the right of well owners to use certain extraordinary means to reduce it to possession. * * * Natural gas is a fluid mineral substance, subterraneous in its origin and location, possessing in a restricted degree the properties of underground waters, and resembling water in some of its habits. Unlike water, it is not generally distributed, and, so far as now understood, it can be used for but few purposes; the most important being that of fuel. Its physical occurrence is in limited quantities only, within circumscribed areas of greater or less extent. * * * but the difference between natural gas and underground waters, whether flowing in channels or percolating the earth, is so marked that the principles which the courts apply to questions relating to the latter are not adapted to the adjustment of the difficulties arising from conflicting interests in this new and peculiar fluid. Natural gas, being confined within limited territorial areas, and being accessible only by means of wells or openings upon the lands underneath which it exists, is not the subject of public rights in the same sense or to the same extent as animals *ferœ naturæ* and the like are said to be. Without the consent of the owner of the land, the public cannot appropriate it, use it, or enjoy any benefit whatever from it. * * * Natural gas in the ground is so far the subject of property rights in the owners of the superincumbent lands, that while each of them has the right to bore or mine for it on his own land, and to use such portion of it as, when left to the natural laws of flowage, may rise in the wells of such owner and into his pipes, no one of the owners of such lands has the rights, without the consent of all the other owners, to induce an unnatural flow into or through his own wells, or to do any act with reference to the common reservoir, and the body of gas therein, injurious to, or calculated to destroy it. Cited in:

No. 18.—Federal Oil Co. vs. Western Oil Co., 121 Fed. Rep. 674.
19.—Richmond Natural Gas Co. vs. Enterprise Natural Gas Co., 66 N. E. 782.

1900

No. 15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., Supreme Court of Indiana, 155 Ind. 545.

"The statute prescribing that it shall be unlawful to conduct natural gas to any point outside of the State is unconstitutional, as affecting interstate commerce, natural gas, when reduced to possession, being an article of commerce, and hence, the Court

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will not enjoin such transportation, no undue appropriation, nor the use of artificial means to produce an unnatural flow from the wells being alleged. * * * the distinction between animals *ferœ naturæ* and natural gas in respect of their ownership before reduction to possession is a very plain one, and has been clearly pointed out in numerous decisions. * * * In the case of wild animals, before they are reduced to possession, the ownership is in the public, and not in any private person; and they are, therefore, held to be subject to the protection of the sovereign. The privilege of taking, killing, and transporting them may, on this ground, be regulated by the legislature. As to natural gas, however, the public has no title to or control over the gas in the ground. On the contrary, so far as it is susceptible of ownership, it belongs to the owners of the superincumbent lands in common, or, at least, such landowners have a limited and qualified ownership in it to the entire exclusion of the public." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N. E. 778.

4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.

Cited in:

No. 19.—Richmond Natural Gas Co. vs. Enterprise Natural Gas Co., 66 N. E. 782.

28.—Haskell vs. Cowham, 187 Fed. Rep. 403.

30.—West vs. Kansas Natural Gas Co., 221 U. S. 229

1900

No. 16.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., Supreme Court of Indiana, 155 Ind. 566.

"In a suit to prevent the transportation of natural gas through pipes at a pressure in excess of the natural rock pressure, and by means other than the natural pressure of the gas flowing from the wells, as prohibited by Burns' Rev. St. 1894, Sec. 7507 (Acts 1891, p. 89), a complaint which does not aver that complainants' property is endangered by the defendant's alleged wrongful acts, nor that they are likely to sustain any special injury peculiar to themselves, is demurrable." Cited in:

No. 19.—Richmond Natural Gas Co. vs. Enterprise Nat. Gas. Co., 66 N. E. 782.

1901

No. 17.—Manufacturer's Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., Supreme Court of Indiana, 156 Ind. 679.

This involved the same demurrable law points as the preceding case. Cited in:

No. 19.—Richmond Nat. Gas Co. vs. Enterprise Natural Gas Co., 66 N. E. 782.

1902

No. 18.—Federal Oil Company vs. Western Oil Company, U. S. Circuit Court of Appeals, 121 Fed. Rep. 674.

"The nature of property in natural gas and oil contained in the earth, and the legal effect of the instrument here in question, have been settled authoritatively by the rulings of the Supreme Court of Indiana." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N. E. 778.

5.—People's Gas Co. vs. Tyner, 131 Ind. 277.

13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

14.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 461.

1903

No. 19.—Richmond Natural Gas Co. vs. Enterprise Natural Gas Co., Appellate Court of Indiana, 66 N. E. 782.

"The Indiana law providing that natural gas shall not be transported through pipes at a pressure exceeding 300 lb. per square inch, nor otherwise than by the natural

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pressure of the gas flowing from the wells, does not prohibit the use of pumps to aid transportation where the pressure is not thereby increased beyond the legal limit, nor does it prohibit the waste of gas; and hence a right in an adjoining owner, tapping a common reservoir, to injunctive relief, is not made out by merely alleging the use of pumps whereby gas is wasted. * * * We cannot agree with counsel for appellees that the legislature, by this statute, has conclusively determined that the effect of the use of pumps is necessarily injurious, and has absolutely prohibited their use. Appellee's case is not made out by merely showing the use of pumps for the purpose of transportation. The use of pumps or compressors for such purpose is not *ipse facto* unlawful. The constitutional validity of the statute is not questioned. * * * It is no longer an open question in this state that natural gas, when reduced to possession, becomes private property, and is a commercial commodity, which the owner may dispose of in whatever manner he may consider most advantageous. * * * Whatever gas, through natural causes, rises to the surface of the ground through the wells of an individual owner, and is carried by such natural forces into tanks, pipe lines, or other receptacles of such owner, becomes absolutely his property. The only limitation upon such owner taking the gas is the manner in which he shall take it from the wells. * * * 'Natural gas in the ground is so far the subject of property rights in the owners of the superincumbent lands that while each of them has the right to bore or mine for it on his own land, and to use such portion of it as, when left to the natural laws of flowage, may rise in the wells of such owner, and into his pipes, no one of the owners of such lands has the right, without the consent of all the other owners, to induce an unnatural flow into or through his own wells, or to do any act with reference to the common reservoir and body of gas therein injurious to or calculated to destroy it.' * * * The sole question here is whether the special findings show that the use of the pumps and compressors will work such injury to appellees, by increasing the flow of gas from the wells, as entitles them to injunctive relief. It is a property of gas, that, while easily compressed, it is always ready to expand and occupy more space. This expansive force or tension imparted to it by some pressure brings it through the wells to the surface, and into pipes or other receptacles. The quantity of the gas that this expansive force or tension, overcoming the atmospheric pressure, will bring to the surface, and into the pipes or receptacles, is the natural flow of the wells. The natural import of the term 'natural flow' as used in the statute, necessarily means the entire volume of gas that will issue from the mouth of a well when retarded only by the atmospheric pressure. If certain wells will bring into a receptacle at the surface 1,000,000 ft. of gas per day, that is their natural flow. Suppose a valve in this receptacle permits one-half this volume to pass into pipes to consumers. Opening an additional valve, and permitting one-half the balance to pass from the receptacle to other consumers, does not increase the natural flow, in the sense in which the term is used in this statute. With the additional valve a greater portion of the natural output of the wells is taken, but the natural flow of the wells has not been increased. And as illustrated in appellant's brief, suppose pumps are attached to a well naturally producing 1,000 bbl. of water per day, for the purpose of forcing 500 bbl. daily to a higher altitude, in order that it may be utilized. Although the water flowed from the well into a main to which the pumps were attached, and the total output put under control, it could not be claimed that the natural flow of the well had been increased by the use of the pumps in carrying half the natural flow to a place for convenient distribution. It must be conceded that the appellant has the right to use all the gas that naturally comes through its wells to the surface of the ground. When we speak of this as the 'natural flow of the wells,' we necessarily mean the total output of the wells from natural causes. The term 'natural flow,' as used in the statute, can be given no other reasonable meaning. * * * Increasing this natural flow of gas or this total output of the wells from natural causes is the thing that is prohibited, and so long as the appli-

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ances take less than this natural flow, as thus understood, it cannot be said that they have increased the natural flow. Upon the facts found, all the gas that goes into the pumps on the intake side goes in by reason of its own expansive force or tension. This expansive force or tension must necessarily exist as long as back pressure is maintained at the pumps. * * * Unless the pumps were so operated as to entirely remove this back pressure and create suction in the wells, it could not be said that they would increase the natural flow or natural output of the wells. From the facts found, the only effect of the use of the pumps is to decrease the force of this back pressure, but this the statute does not prohibit. It is true that through the use of the pumps a greater portion of the natural flow or total natural output of the wells will be taken than would be taken without their use, and that the back pressure in the wells will be correspondingly decreased. * * * It cannot be denied that appellant might lawfully dispose of the total product of its wells to consumers near the wells, and the total natural flow be consumed by them, without maintaining any back pressure in the wells. * * * While the court found, as an ultimate fact, that the use of the pumps has the effect of increasing the general flow of gas from the wells, it also found the primary facts from which it appears that the use of the pumps would not increase the amount of gas coming from the wells through the natural laws of flowage. In such case the ultimate fact will be disregarded on appeal." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N. E. 778.

4.—Jamieson vs. Indiana Nat. Gas & Oil Co., 128 Ind. 555.

13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

14.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 461.

15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.

16.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 566.

17.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 156 Ind. 679.

1903

No. 20.—Consumers' Gas Trust Co. vs. American Plate Glass Co., Supreme Court of Indiana, 68 N. E. 1020.

"The mere fact that the complaint disclosed that the gas company was using a pumping station in the winter time, at a point some 6 miles from the land in question, for the purpose of overcoming the friction incident to the flow of gas through many miles of pipe, does not stamp such company as a lawbreaker. There is nothing to suggest that by means of such pumping station the natural pressure of the gas at the wells was increased, or that the pressure in the pipe exceeded 300 lb. per square inch."

1903

No. 21.—Acme Oil & Mining Co. vs. Williams, 74 Pac. Rep. 296.

"leases are only valuable on development, and are then only valuable to both parties to the extent that the product may be secured and disposed of; and when the only consideration for the lease is the share which the lessor will obtain of what is produced, there is always an implied covenant that diligence will be used toward such production. There are few other mining enterprises where delay is so dangerous, and where diligence in securing immediate possession of the mineral is so necessary, as is mining for oil. As to the precious metals, fixed in the veins which hold them, they remain intact until extracted. Oil, on the contrary, is of a fluctuating, uncertain, fugitive nature, lies at unknown depths, and the quantity, extent and trend of its flow are uncertain. It requires but a small surface area, in what is known as an oil district,

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upon which to commence operations for its discovery. But when a well is developed the oil may be tributary to it for a long distance through the strata which holds it. This flow is not inexhaustible, no certain control over it can be exercised, and its actual possession can only be obtained, as against others in the same field, engaged in the same enterprise, by diligent and continuous pumping. It is anybody's property who can acquire the surface right to bore for it, and when the flow is penetrated, he who operates his well most diligently obtains the greatest benefit, and this advantage is increased in proportion as his neighbor similarly situated neglects his opportunity."

1905

No. 22.—Brewster vs. Lanyon Zinc Co., U. S Circuit Court of Appeals, 140 Fed. Rep. 801.

"Experience in other oil and gas fields has demonstrated that wells drilled in the vicinity of producing wells were not infrequently unproductive. The only method of certainly determining whether or not particular lands contained oil or gas in paying quantity was by drilling thereon to considerable depth. * * * Light will be thrown upon the language used, and the intention of the parties will be better reflected, if consideration is given to the peculiar and distinctive features of the mineral deposits which are the subjects of the lease. Oil and gas are usually found in porous rock at considerable depth under the surface of the earth. Unlike coal, iron, and other minerals, they do not have a fixed situs under a particular portion of the surface, but are capable of flowing from place to place and of being drawn off by wells penetrating their natural reservoir at any point. They are part of the land, and belong to the owner so long as they are in it, or are subject to his control; but when they flow elsewhere, or are brought within the control of another by being drawn off through wells drilled in other land, the title of the former owner is gone. So, also, when one owner of the surface overlying the common reservoir exercises his right to extract them, the supply as to which other owners of the surface must exercise their rights, if at all, is proportionally diminished. * * * By reason of the conditions on which the lease is granted the lessor retains at least a contingent interest in the oil and gas, to the profitable extraction of which the operations are directed. This interest in the subject of the lease, and the fact that the substantial consideration for the grant lies in the provisions for the payment of royalty in kind and in money on the oil and gas extracted, make the extent to which and the diligence with which the operations are prosecuted of immediate concern to the lessor. If they do not proceed with reasonable diligence, and by reason thereof the oil and gas are diminished or exhausted through the operation of wells on adjoining lands the lessor loses, not only royalties to which he would otherwise be entitled, but also his contingent interest in the oil and gas which thus passes into the control of others." Cites:

No. 8.—Brown vs. Spilman, 155 U. S. 665.

13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

1908

No. 23.—Calor Oil & Gas Co. vs. Franzell *et al.*, Court of Appeals of Kentucky, 109 S. W. Rep. 331.

"One who illegitimately wastes or destroys the gas of a district may be punished under the criminal statutes of the State, and may also be enjoined from committing such wrongful acts; but all parties owning gas wells in the district are free to make any legitimate use of the gas they choose, and the fact that this legitimate use tends to exhaust the supply gives the other owners of gas wells in the district no just ground of complaint. The Court knows, as a part of the history of the country, that natural gas districts after flourishing for a while are frequently entirely exhausted, and that manufacturing and power plants established in a district, and dependent upon the use of

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the gas for fuel, are forced to move elsewhere. This may happen to the district under consideration; but, as said before, if this results from legitimate sales by the various owners to other customers, no one of them has a just ground for complaint. They have all enjoyed the property while it existed, and when it is exhausted they, of course, can no longer enjoy it."

1908

No. 24.—Louisville Gas Co. *et al.* vs. Kentucky Heating Co., Court of Appeals of Kentucky, 111 S. W. Rep. 376.

"The right of the surface owners to take gas from subjacent fields or reservoirs is a right in *common*. There is no property in the gas until it is taken. Before it is taken it is fugitive in its nature, and belongs in common to the owners of the surface. The right of the owners to take it is without stint; the one limitation being that it must be taken for a lawful purpose and in a reasonable manner."

1909

No. 25.—Kansas Natural Gas Co. vs. Haskell, United States Circuit Court, 172 Fed. Rep. 545.

"From all of which it becomes apparent the contention of defendants that the natural gas found within the territorial limits of the State is the common heritage of the people of the State, which may be conserved and preserved by the State as trustee of those things in which the people have a common interest, as flowing streams, wild animal life, etc., is unsound and must be denied. On the contrary, it must be held he who by lawful right reduces to his possession mineral, gas, or oil, has the same absolute right of property therein, with the same power of barter, sale, or other disposition, including, of necessity, the right of transportation and delivery under such reasonable rules and safeguards as the exigencies of the case may demand and the State employ, as the farmer has of his corn, his wheat, or his stock, or the merchant of his wares, and such absolute right therein as the State cannot deny him without making just compensation, and any attempt to do so would be in violation of the fourteenth amendment to the federal Constitution." Cites:

No. 3.—Westmoreland, etc., Co. vs. DeWitt, 130 Pa. 235.
4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.
8.—Brown vs. Spilman, 155 U. S. 665.
10.—Thomas C. Kelley vs. Ohio Oil Co., 57 Ohio State, 317.
11.—State of Indiana vs. Ohio Oil Co., 150 Ind. 21.
13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

Cited in:

No. 30.—West vs. Kansas Nat. Gas Co., 221 U. S. 229.

1910

No. 26.—Ilo Oil Co. vs. Indiana Natural Gas & Oil Co., 174 Ind. 635. Indiana Supreme Court.

"A plaintiff that is pumping oil by the use of artificial means has no standing in court to enjoin defendant from doing the same thing, but in a greater degree, since the plaintiff must do equity and come into court with clean hands."

1910

No. 27.—Kolachny vs. Galbreath, 26 Oklahoma 772.

"Oil and gas, while in the earth, unlike solid minerals, are not the subject of ownership distinct from the soil, and the grant of the oil and gas, therefore, is a grant, not of the oil that is in the ground, but of such part as the grantee may find and passes nothing that can be the subject of an ejectment or other real action." Cited in:

No. 29.—Frank Oil Co. vs. Bellview Gas & Oil Co., 29 Okla. 719.

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1911

No. 28.—Haskell vs. Cowham, United States Circuit Court, 187 Fed. Rep. 403.

"The right of a private citizen by means of his ownership of, or of his mining leases on, land to draw gas or oil from beneath its surface is property and sometimes valuable property. * * * Interstate commerce in natural gas, including the transportation thereof in pipe lines, is a subject national in its character and susceptible of regulation by uniform rules, and the silence or inaction of Congress regarding it is a conclusive indication that it intends that interstate commerce therein shall be free and laws of States, and acts of its officers which either prohibit or substantially burden it are without justification violative of the Constitution and void. * * * Neither a State nor its officers may prevent or unreasonably burden interstate commerce in the natural gas within it by preventing the use to transport it of pipe lines across the highways of the State by means of the exercise, or the refusal to exercise, the police powers or the proprietary powers of the State over them." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N. E. 778.

13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

15.—Manufacturers' Gas & Oil Co., vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.

1911

No. 29.—Frank Oil Co. vs. Bellview Gas & Oil Co., 29 Okla. 719.

"Oil and gas, while in the earth, unlike solid minerals, are not the subject of ownership distinct from the soil, and the grant of the oil and gas, therefore, is a grant not of the oil that is in the ground, but of such part as the grantee may find, and passes nothing except the right to explore for the same under the terms of such contract. * * * A different rule of construction obtains as to oil and gas leases from that applied to ordinary leases or to other mining leases. Owing to the peculiar nature of the mineral, and the danger of loss to the owner from drainage by surrounding wells, such leases are construed most strongly against the lessee and in favor of the lessor. Cites:

No. 27.—Kolachny vs. Galbreath *et al.*, 29 Okla. 772.

1911

No. 30.—West vs. Kansas Natural Gas Co., United States Supreme Court, 221 U. S. 229.

"The case is a valuable one and clearly announces the rights of an owner to the soil beneath it, and the relation of his rights to all other owners of the surface of the soil. The right of taking the gas, it was said, was common to all owners of the surface, and because of such a common right in all land owners, and unlimited use (against a wasteful use the statute was directed) by any it was competent for the State to prohibit. This limitation upon the surface owners of property was justified by the peculiar character of gas and oil, they having the power of self-transmission, and that therefore to preserve an equal right in all surface owners there could not be an unlimited right in any. Gas and oil were likened to, not made identical with, animals *ferœ naturæ*, and, like such animals, were subject to appropriation by the owners of the soil, but also, like them, did not become property until reduced to actual possession. But an important distinction was pointed out. In things *ferœ naturæ*, it was observed, all were endowed with the power of reducing them to possession and exclusive possession. In the case of natural gas, only the surface proprietors had such power, and the distinction, it was said, marked the difference in the extent of the State's control. In the one, as the public are the owners, everyone may be absolutely prevented from seeking to reduce to possession. No devesting of private property, under such a condition, can be conceived, because the public are the owners, and the enactment by the State

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of a law as to the public ownership is but the discharge of the governmental trust resting in the State as to property of that character. * * * On the other hand, as to gas and oil, the surface proprietors within the gas field all have the right to reduce to possession the gas and oil beneath. They could not be absolutely deprived of this right which belongs to them without a taking of private property.' And this right, it was further said, was coequal in all of the owners of the surface, and that the power of the State could be exerted 'for the purpose of protecting all the collective owners, by securing a just distribution, to arise from the enjoyment by them of their privilege to reduce to possession, and to reach the like end by preventing waste.' And further characterizing the statute, it was said, viewed as one to prevent the waste of the common property of 'the surface owners, it protected their property, not devested them of it. And special emphasis was given to this conclusion by the comment that, to assert that the right of the surface owners to take was, under the 14th Amendment, a right to waste, was to say that one common owner may devest all the others of their rights without wrongdoing; but the lawmaking power cannot protect all the owners in their enjoyment without violating the Constitution of the United States.' The case, therefore, is an authority against, not in support of, the contention of the appellant in the case at bar. The statute of Indiana was directed against waste of the gas, and was sustained because it protected the use of all the surface owners against the waste of any. The statute was one of true conservation, securing the rights of property, not impairing them. Its purpose was to secure to the common owners of the gas a proportionate acquisition of it—a reduction to possession and property—not to take away any right of use or disposition after it had thus become property. It was sustained because such was its purpose. * * * Gas, when reduced to possession, is a commodity; it belongs to the owner of the land; and, when reduced to possession is his individual property, subject to sale by him, and may be a subject of intrastate commerce and interstate commerce. * * * These propositions were announced: (1) Natural gas is as much a commodity as iron ore, coal, or petroleum, or other products of the earth, and can be transported, bought and sold as other products. (2) It is not a commercial product when it is in the earth, but becomes so when brought to the surface and placed in pipes for transportation." Cites:

No. 2.—State vs. Indiana & Ohio Oil, Gas & Mining Co., 22 N.E. 778.

No. 4.—Jamieson vs. Indiana Natural Gas & Oil Co., 128 Ind. 555.

No. 13.—Ohio Oil Co. vs. Indiana, 177 U. S. 190.

No. 15.—Manufacturers' Gas & Oil Co. vs. Indiana Natural Gas & Oil Co., 155 Ind. 545.

No. 25.—Kansas Natural Gas Co. vs. Haskell, 172 Fed. Rep. 545.

Cited in:

No. 31.—Haskell vs. Kansas Natural Gas Co., 224 U. S. 217.

1912

No. 31.—Haskell vs. Kansas Natural Gas Co., United States Supreme Court, 224 U. S. 217.

"Natural gas after severance is a commodity which might be dealt in like other products of the earth, as coal and other minerals, and is a legitimate subject of interstate commerce; and that no State, by such laws as were involved in the case, can prohibit its transportation in interstate commerce beyond the lines of that State, The court held, after considering and construing the provisions of the act of 1907, that it was, upon its face, a law undertaking to prohibit the transmission or transportation in interstate commerce of natural gas to points beyond the State; that it was an unconstitutional interference with the rights of the complainants, who were legitimately engaged in that commerce, and that therefore the act was null and void." Cites:

No. 30.—West vs. Kansas Natural Gas Co., 221 U. S. 229.

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Discussion

David T. Day, Washington, D. C.—I would like to ask whether the court decisions quoted in the paper are general, or do they represent one side of the case; if so, are there any decisions that represent more than one side of the case? Are they of general application, or do they affect only one side?

Also, what is the difference between a pump and a compressor from the standpoint of the natural-gas industry? I notice stress is laid on the word "pump" in connection with some of the legal decisions, and then it is stated that they must not pump or compress above 300 lb. The question is, is a pump a compressor and a compressor a pump, under certain conditions?

Samuel S. Wyer, Columbus, Ohio.—Answering the first question, I will say that the decisions cover both sides of the case very thoroughly, and are uniformly on the side that a company is justified in the use of gas compressors; in other words, there are no decisions holding it to be illegal to use gas compressors, with the exception noted in Indiana, Kansas, and Oklahoma where you cannot go below atmospheric pressure.

In regard to the second question, I think the engineers and the gas operators themselves are responsible for the confusion. It is because the use of the term "pump" is almost universal, and so many people have obtained the idea that the primary function of a gas compressor is to pump air into the gas, in order to increase the consumers' gas bills.

I appreciate that it is very hard to draw a clear line of demarcation between a pump and a compressor. The distinction I have always made myself, and which I think is a logical one, is that you do not get the pumping action until you get below atmospheric pressure.

As far as the compression end is concerned, there is only one State, namely, Indiana, that has any statutory limitation with regard to how high the pressure may be increased in order to put the gas into the transmission lines. The State law in Indiana limits the maximum pressure to 300 lb. per square inch.

I. N. Knapp, Ardmore, Pa.—I think the way the confusion in terms between a pump and a compressor originated was that years ago when gas compressing was first attempted it was done with a vertical pump cylinder placed at one well and operated from the walking beam. This was called gas pumping. Afterward when several wells by long suction lines were attached to a regular compressor, this was called gas compressing.

W. L. Saunders, New York, N. Y.—I am rather surprised that there should be any question as to the difference between a pump and a compressor in gas compression. A compressor may become a pump at any

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time, if it is so applied to its work that it is expected to suck from a well or from a retort, where the pressure may be less than the atmosphere, or greater than the atmosphere, and then, on the other side of the piston, it becomes a compressor; in other words, it is common practice in the manufacture of machinery for gas compression and gas transmission to use practically the same apparatus, both for pumping (sucking) and for compressing (raising the pressure to a higher degree and discharging it into a line or into a retort).

DAVID T. DAY.—Is it a fact that a State Legislature has taken cognizance of the idea that air is pumped in with natural gas, to include in their definition of a pump something that takes air and pumps it into natural gas?

SAMUEL S. WYER.—Not to my knowledge.

I. N. KNAPP.—In a gas case in Omaha, Neb., the complainant alleged that air was pumped into the gas. By expert testimony it was established that if air was pumped into this manufactured gas, as alleged, its illuminating value would be destroyed and so it would have no merchantable value. The Judge of the Court discredited this testimony by saying, "I believe this gas company does put air in their gas and they do sell God's free air for $1.35 per foot."

This is not an unusual exhibition of the "judicial mind" in gas cases.

The gas rate was $1.35 per thousand cubic feet.

W. L. SAUNDERS.—Will Mr. Wyer explain why in Indiana, or anywhere else, the pressure should be limited to 300 lb? It is not uncommon to carry compressed air to 2,000 or 3,000 lb. pressure. I fail to see why there should be a limitation. The strength of the vessel containing the compressed air should be the limit of the pressure, and the vessel should be properly inspected. I fail to see why there should be any distinction in the matter of gas.

SAMUEL S. WYER.—In passing a law like that, the Supreme Court of the United States, upholding the authority of the Legislature, under the general police power, really answered the question by sustaining the power of the legislature to fix some limit which would be safe, but refused absolutely to say that 300 lb. per square inch was a safe figure to use in that case. That law, of course, was passed a number of years ago in the early days of the art, and the people at that time thought if they had more than 300 lb. pressure in the line it became unsafe, and, therefore, under the general police powers, it was held that the State had the right to regulate it. At the present time a number of lines in other States are operating at about 400 lb. pressure.

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HARRISON SOUDER, Cornwall, Pa.—I would like to ask why the small producers object to the larger producers having a compressor or pump on the line.

SAMUEL S. WYER.—There are two reasons for their objections. In the first place it is generally purely a blackmailing proposition. The small producer, upon finding that the production is going down and he is unable to make money, in many cases takes one last chance by blackmailing the fellow who has been successful.

The other reason is the misapprehension as to the ownership of gas. Many men do not appreciate that there is no ownership in gas until it is reduced to possession. In other words, there is not a single conflicting opinion on the point that there is absolutely no property right and no ownership of the gas under the realty until that gas has been reduced to possession, so that it can come under the owner's control.

I. N. KNAPP.—Mr. Wyer speaks of the little gas producer blackmailing the large producer. Now I would like to present the other side of that question.

I have had a large producer with a market for his gas come to me when I was a small oil producer and say: "Well; Knapp you are in the oil business and we hear you have found some pretty good gas wells. Now, as an oil man you can't have much use for all the gas you have; those wells must be sort of dead stock on your hands. How would you like to get back the money the gas wells cost you?"

"Well," said I, "if I drilled a lot of dry holes you would not come around here asking me how I would like to get my money back. My gas wells are worth several times their cost."

"Oh, well, you cannot sell the gas. It is not worth anything to you. You are mighty lucky to get the offer we make. If you do not accept we will drill around here on our leases and get all the gas anyway."

This shows the other side of the question.

F. G. CLAPP, New York City.—I would like to ask Mr. Wyer regarding the laws in fields where the sands are wet as distinguished from those where they are dry.

SAMUEL S. WYER.—There is no law on the subject, outside of the recent conservation order of the Oklahoma Corporation Commission.

I. N. KNAPP.—I believe Mr. Clapp asked a question about water raising in a gas sand. I once operated in a small field where the productive gas sand appeared to be about ¼ mile wide by 1½ miles long. The sand was 10 to 15 ft. thick, somewhat like the Berea grit, only softer. The State Geologist considered this pool was in the bottom of a synclinal trough. There were five or six wells in it.

If you drew too strong on any one well salt water would appear very

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promptly. I imagine the water drew up to a cone shape in the sand with the apex entering the well. By shutting off such a well for a few days the water disappeared or rather seemed to level off.

The best results indicated that the wells should all be drawn on evenly. The top of the sand was quite level as near as could be measured with a steel line and anticlinal conditions cut no figure at all.

F. G. CLAPP.—It would seem pertinent here to mention the discrepancy between fields where the sands are wet, or saturated with water, and fields where the sands are dry. It is a fact that the Clinton sand of Ohio, which is so full of gas, has scarcely any water in it, while, on the other hand, in the Oklahoma, Kansas, and Texas fields, the sands are saturated with water.

It is a question whether the law should apply with the same weight in fields of the two types. For instance, when a field is being developed in Ohio, gas can be drawn out with or without compressors, and no matter how hard it is drawn upon, no water will enter the space surrounding the well, to take the place of the gas exhausted, unless, of course, the casing be faulty. On the other hand, if we suppose a common instance, where an individual or a company opens a well on the down-dip side of the field, and allows the gas to flow out under its own pressure, the pressure is caused to decline and the water at once commences to encroach on that side of the field. If the gas companies are using compressors in that pool the encroachment of water is much more active. Should not the laws, in all cases, be worded so as to distinguish carefully between the use of compressors in fields where the sands are wet and those where they are dry; also to distinguish between local use of compressors in a single well or group of wells and their more general use through the entire field? I would like to ask Mr. Wyer if it is not desirable to enact laws which will distinguish between the different classes of fields and different phases of compressor use?

SAMUEL S. WYER.—My own idea, based on the development of the common law principles, as given in the paper, is that there ought to be no restriction at all with regard to the removal of the gas; in other words, every foot of gas in the ground that can be removed ought to be, and the only way that it can be thoroughly removed, in practically all cases, is by the use of gas compressors.

Answering the other phase of your question, in this particular case of litigation, the producer who brought the suit operated about 70 wells in the Ohio field, and he had only one market, namely, Mount Vernon, Ohio. The wells were scattered all around the town, and he had three main lines from these wells going into that one market, and therefore his length of haul for his own gas was very much shorter than the large company that had markets all over the State of Ohio.

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That made such a marked difference that, as stated in the last paragraph, out of 4,510 hourly pressure comparisons, the small producer, without any compressor 90 per cent. of the time, was maintaining lower pressures than prevailed on the adjacent wells which discharged into the intake of the large compressing station, and 25 per cent. of the time he had a lower pressure on those wells than was maintained at the intake of the station, I mean right within a few feet of the intake side of the compressor. That shows that if market conditions are favorable, in many instances the pressure can be reduced very much lower merely by large lines of short length than is possible even with a large compressing station.

F. G. CLAPP.—If we should consider that matter a little further, I presume we would all agree regarding dry sands, since no harm can come to the field, no matter how strongly the gas is drawn out with a compressor, but there still remains the question of a wet sand, as in Texas and Oklahoma. As Mr. Wyer says, a company should be allowed, for the good of the public and itself, as much gas as can be drawn out. But then I would raise the question, if a company enters a wet field and pulls too strongly on part of it, whether as much gas can be drawn out as if it were taken out through a slower process.

In certain fields which I have in mind, particularly a long, narrow one in Texas, like Mr. Knapp has mentioned, the field is about 15 miles long and a little over ½ mile wide. The salt water saturates the sand along the eastern side, and in a very few months the wells drawn upon along that side have been flooded by water, necessitating the abandonment of some of them. It is a serious question whether there should not be some regulation, not one to hamper the companies, but some sort of regulation or provision so that an engineer placed in charge of the entire field should insist that the gas be taken out uniformly and scientifically to prevent any drowning by water.

This seems to me a very important subject, as so many fields have been ruined by influx of water. Probably that, more than anything else, has caused the failure of so many of our good gas fields early in their history.

We would presumably all agree that whatever laws be enacted, or whatever regulation be made, should be for the best interest of the companies, especially the public utility companies, because they are, as a rule, honestly doing their best to supply the people with gas, and my opinion is that no gas field, no matter how large it is, whether it be the central Ohio field, or one in Texas, should be operated by more than one company. In my mind, a gas field can be operated to best advantage and for the greatest good of a company and its consumers only if the company is unrestricted in developing the particular field in which it is operating.

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In the gas business we may say, as opposed to our ideas in many lines of business, that we believe in monopoly. We know that the large gas companies understand their business, they know how to take out the gas, how to sell it, and most of them know what the rates should be. While we can wisely restrict them in certain particulars, we should allow them as much freedom as possible in developing the field for the best interests of the people.

That, I think, has been done by the large gas companies which are operating in Pennsylvania, West Virginia, and Ohio, and which have generally developed their fields wisely, operating individual fields for many years, and practically all these communities are still being supplied with gas. This is largely due to the moderate degree of competition. But in Oklahoma, on the other hand, sometimes half a dozen or more companies are located in and around a single section, and several wells are drilled within a radius of a few thousand feet, which is very bad for the business, resulting in a great economic loss. Sooner or later one of the companies is obliged to jump in with a compressor, then the others follow suit, and the field is rapidly drowned through influx of water. If that development were carried on through the operations of a single company, however, it would be done more wisely, and in a scientific manner, rather than in a haphazard way, as is the case where a number of individuals or companies have control.

I must return to the point first mentioned, and still raise the question whether the law should take cognizance of the fact that some fields are wet and some are dry, so as to require the companies developing the wet fields to do so at the best possible advantage.

SAMUEL S. WYER.—In answer to Mr. Clapp's suggestion, I will say it is physically impossible to do that, on account of the manifestly unfair attitude of the United States Department of Justice. For four years the Department has been hounding the companies (I happen to be familiar with the facts) with a view to trying to get them to stop doing the very things Mr. Clapp has recommended, in other words, working in harmony and in unison in the development of the fields.

According to the Department of Justice, the public is not well served unless there is cut-throat competition in the field. That, as any one knows, is of course decidedly wrong, and until such time as the United States Department of Justice turns about face and works in harmony with the Bureau of Mines and the Geological Survey on conservation matters it will not be possible to get unified control conditions in the natural gas fields that are absolutely essential to true conservation and to the best development of our natural resources.

The attitude of the Department in preventing regulated, unified control has cost the public many thousands of dollars in the depletion of fields that could have been maintained for a large number of years

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had they been properly operated, which operation, however, could have been properly continued only by an absolute unified control of the entire field, such as Mr. Clapp suggests. This, however, is the very thing the agents of the United States Department of Justice have tried to prevent at all times up until the present time, as well as at the present time.

W. L. SAUNDERS.—I would like to ask Mr. Wyer a question. Has this subject been brought to the attention of the Federal Trade Commission? The Federal Trade Commission is a mediator between the public and the Department of Justice, and in many things similar to this the Federal Trade Commission has been very helpful to the business interests, by hearing the case, giving special advice, and avoiding contact with the Department of Justice in certain cases.

We must bear in mind that the Department of Justice is created to enforce the law; it has no authority except to enforce the law. It cannot change the law. It simply must tell you that the law (in these cases I presume it is the Sherman Law), goes contrary to what you are doing, and you must stop, but the Sherman Law is not a law like that of the Medes and Persians. The Sherman Law is capable of a good deal of elasticity, and since the establishment of the Federal Trade Commission, the business men in various lines have found it advantageous and very profitable in the interest of good business to coöperate with the Federal Trade Commission in avoiding contact with the Department of Justice.

I should like further to ask Mr. Wyer if this trouble with the Department is a recent or an old one?

SAMUEL S. WYER.—Answering the two inquiries in the order in which they were made, so far as I know, no attempt has been made to secure relief through the new Commission. I believe, however, there is an opportunity there of bringing about adequate relief as Mr. Saunders suggests.

With regard to the second query, the trouble is not a new one; it has been going on ever since the Department became active in looking up the alleged unification of interests in the gas business, which, of course, dates back a number of years.

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Necessary Use and Effect of Gas Compressors on Natural Gas Field Operating Conditions*

BY SAMUEL S. WYER,† M. E., COLUMBUS, OHIO

(New York Meeting, February, 1916)

1. THE following is an abridgment of a recent report made by the author, covering an investigation of:

(A) The necessary use of natural-gas compressors;

(B) The effect of gas compressors on natural-gas field operating conditions in Ohio, with special reference to the working pressure relations of gas wells of one company discharging by natural rock-pressure flow without compressors, and adjacent wells of other companies, discharging into intake lines to gas-compressing stations.

Tests were made at 107 different places in the counties of Fairfield, Licking and Knox, Ohio, which required 7,000 miles of automobile driving, and consumed 6 months' time.

2. A proper conception of the necessary use and effect of natural-gas compressors in natural-gas production necessitates a clear understanding of certain fundamental definitions and judicial doctrines, which are given herein.

3. Gas is a fluid composed of a large number of molecules which are vehicles of energy continually in motion, and having an inherent tendency to get farther and farther apart. The range of motion of the molecules is limited only by the volume of the closed containing vessel in which they constantly move to and fro. The most distinguishing characteristic of gas is its universal property of completely filling an inclosed space.

4. Gas pressure is the result of the combined efforts of all the moving molecules in the mass trying to get farther and farther apart, that is, a mass of gas inclosed in a vessel expands and fills it, and being restrained from further expansion, it exercises a pressure against the walls of the vessel. This pressure is the same in all directions on equal areas of surface, as shown in Fig. 1. With a given mass of gas, any increase in volume of containing vessel will give the molecules more range of motion,

* The public's interest in the age and magnitude of the art of gas compressing and development of the law relating to the use of gas compressors in natural gas production is discussed in the preceding paper, Development of the Law Relating to the use of Gas Compressors in Natural Gas Production; see also Is It Feasible to Make Common Carriers of Natural Gas Transmission Lines, vol. xlviii, 471 to 480 (1914).

† Consulting Engineer.

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and thereby lower the pressure. Thus, if a part of a given mass is removed from a closed vessel or reservoir, the remaining mass of gas will expand instantly and keep the vessel or reservoir *filled*, but at a *lower* pressure.

5. The pressure per unit of area exercised inward upon a mass of fluid is transmitted undiminished in all directions, and acts with the same force upon all surfaces in a direction at right angles to those surfaces. Hence, the pressure applied to any area of a confined fluid is transmitted to every other equal area through all the fluid, to the walls of the containing chamber, without diminution. This is known as Pascal's law, and is shown in Fig. 1. According to this, the gas pressures in the various parts of a

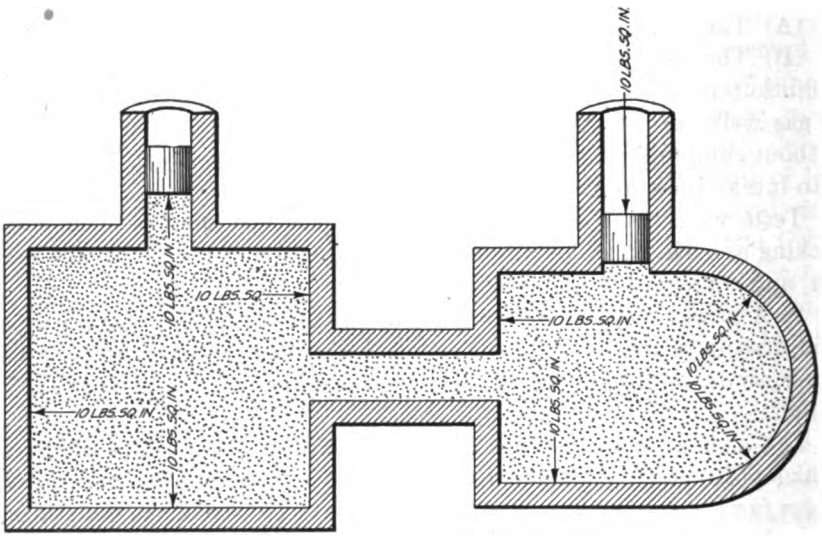

FIG. 1.—DIAGRAM ILLUSTRATING PASCAL'S LAW.

"continuous and connected reservoir" must be equal. If they are not equal, the various parts of the reservoir are not continuous or connected.

6. As long as energy of any form undergoes no transformation it tends to gravitate to a lower degree of intensity, that is, become more stable and approach a universal level of stable equilibrium. Without external assistance fluids can flow only from higher to lower levels. Thus, water always seeks the lowest level, and confined gas tends to expand to lower pressures, and gases confined in "connected or continuous reservoirs" always tend to and must ultimately equalize their pressures, and gas flow in pipes or underground reservoirs is always in the direction of the point of lower pressure.

7. Boyle's law is "The volume of gas is inversely proportional to the absolute pressure to which the gas is subjected; or, what is the same thing, the product of the absolute pressure and the volume of a given quantity

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of gas remains constant." This law is not followed literally by natural gas, but various gases deviate from it slightly, varying with the composition of the gas itself, and the pressure to which the gas is subjected.[1]

8. The word "barometer" means instrument for measuring weight, and it indicates merely the pressure of the air at the point where it is placed. Sea level is the datum from which atmospheric pressures are reckoned. At that point dry air at 32°F. exercises a pressure of 14.7 lb. per square inch. The relation of atmospheric pressure to the other pressure terms is shown in Fig. 2.

9. The term "vacuum" has three distinct meanings.

(A) Literally, a space entirely devoid of matter. This is merely a theoretical conception.

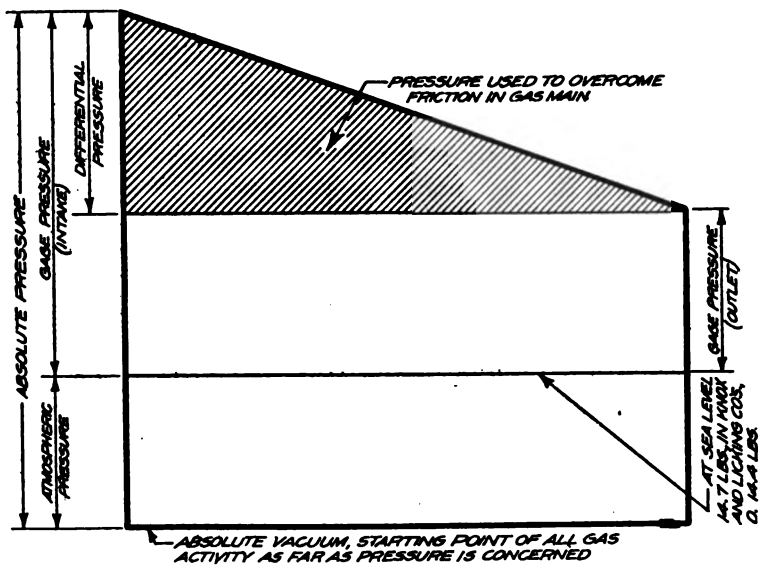

FIG. 2.—DIAGRAM ILLUSTRATING GAS-PRESSURE TERMS.

(B) A space nearly devoid of matter, or a vessel from which the air has been exhausted to a very high degree, as may be done by apparatus in a scientific laboratory. Such high exhaustions are called by courtesy, vacua, as they are the nearest approaches that scientists have been able to make to absolute vacuum by the most refined methods known to science, and yet they are far from an absolute vacuum.

(C) Ordinary use of the term means merely a partial diminution of pressure below the normal atmospheric pressure. This is the engineering conception, and the maximum attainable degree with ordinary engineer-

[1] This deviation is discussed in detail in a paper presented at the Spring Meeting (April, 1916) of the American Society of Mechanical Engineers.

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NAME OF WELL	FRANK WHITE	WILMOT SPERRY	ELLEN M. JACKSON	W. L. PARROT	G.R. GEORGE	D.W WALLACE	ELIZABETH KAISER	CHARLES CAREY	W.C. COOPER	W.O. HAWKINS
TOWNSHIP	COLLEGE	CLINTON	MILFORD	CLINTON	CLINTON	MORGAN	CLINTON	LIBERTY	CLINTON	CLINTON
DATE	NOV. 28	NOV. 21	NOV. 27	DEC. 22	NOV. 20	DEC. 22	NOV. 22	NOV. 26	DEC. 21	DEC. 21
GAGE PRESSURE AT WELL	35 LBS.	17 LBS.	22 LBS.	14 LBS.	20 LBS.	10 LBS.	17 LBS.	15 LBS.	16 LBS.	16 LBS.
GAGE PRESSURE AT COMPRESSOR INTAKE	8 LBS.	4 LBS.	14 LBS.	1 LB.	5 LBS.	1 LB.	4 LBS.	8 LBS.	5 LBS.	5 LBS.
PRESSURE DROP BETWEEN WELL AND COMPRESSOR	27 LBS.	13 LBS.	8 LBS.	13 LBS.	15 LBS.	9 LBS.	13 LBS.	7 LBS.	11 LBS.	11 LBS.
NATURAL OR OPEN FLOW IN 24 HOURS Cu. FT.	49500	45000	132000	75000	70000	74000	50000	49500	100000	87000
ACTUAL FLOW INTO LINE IN 24 HOURS Cu. FT.	18000	24000	76800	45600	43000	48000	33600	33600	88000	76800
PERCENT OF OPEN NATURAL FLOW USED	32	53	58	60	61	64	67	68	88	88

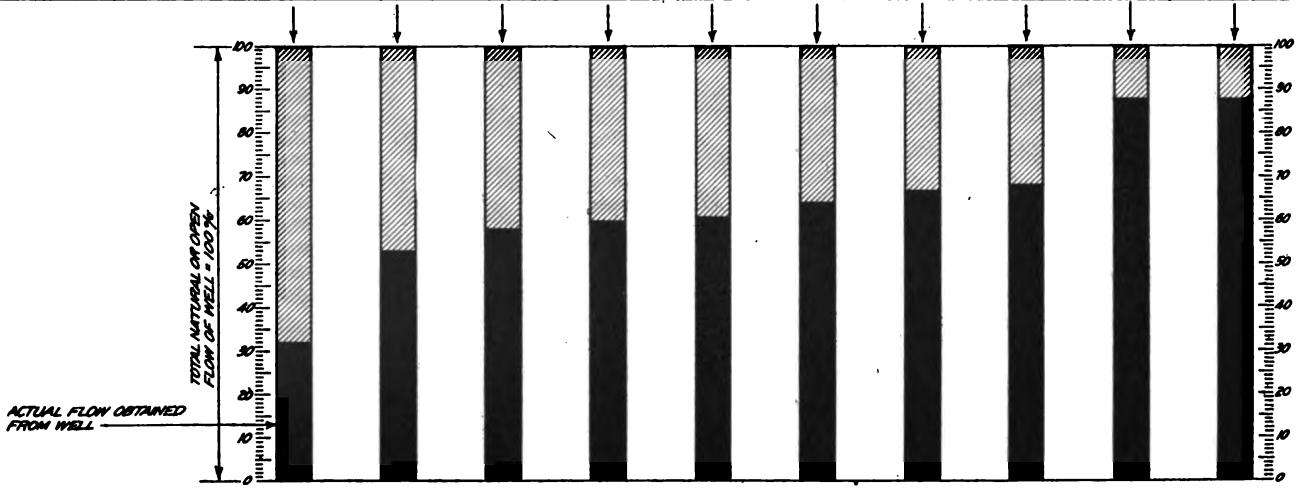

FIG. 9.—NATURAL OR OPEN FLOW, AND ACTUAL LINE FLOW FROM VARIOUS WELLS OF OHIO FUEL SUPPLY CO. IN KNOX COUNTY, OHIO, IN 1913, DISCHARGING INTO INTAKE LINES TO HOMER GAS-COMPRESSING STATION.

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absolute-pressure scale on the right-hand side, while the dates are always marked at the bottom.

31. The observations and data obtained led to the following conclusions:

(A) Ordinarily only a relatively small percentage of the natural flow may be obtained from a natural-gas well under routine operating condi-

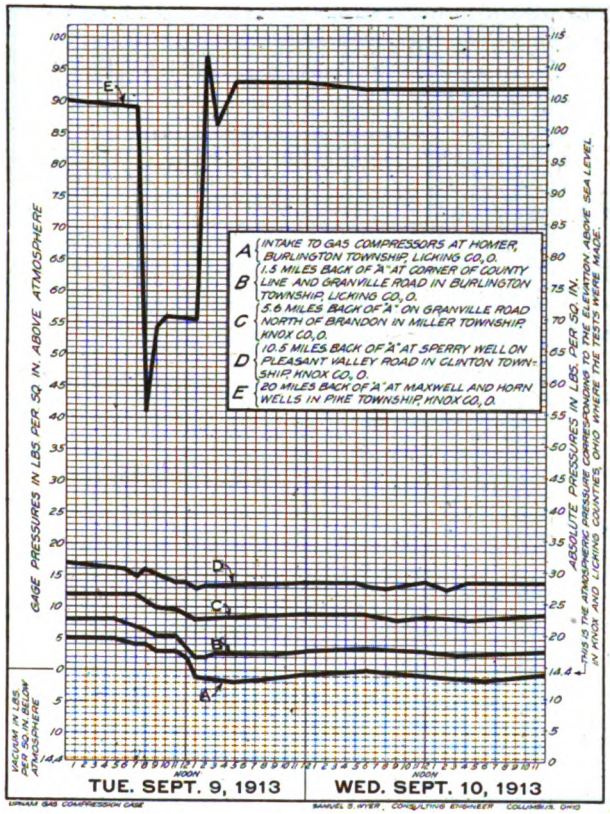

FIG. 10.—SHORT RANGE OF LOW PRESSURES IN "MILLER INTAKE LINE" OF OHIO FUEL SUPPLY CO., DISCHARGING IN HOMER COMPRESSING STATION, BASED ON SIMULTANEOUS RECORDING PRESSURE-GAGE RECORDS.

tions, even when such a well is discharged into the intake line to a gas-compressing station.

(B) That this Ohio field is not made up of one large "connected and continuous reservoir," but rather composed of many separate and discontinuous reservoirs, is evident from:

(*a*) The large number of dry holes drilled.

(*b*) The large number of wells that have been abandoned in close proximity to producing wells.

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(*c*) The continuous difference in rock pressure in adjacent wells that have been shut in for a period of years.

(*d*) The marked variation in working pressures of adjacent wells.

(*e*) The marked variation in rock pressure in different parts of the field.

(C) The diminution in rock pressure has simply been the normal

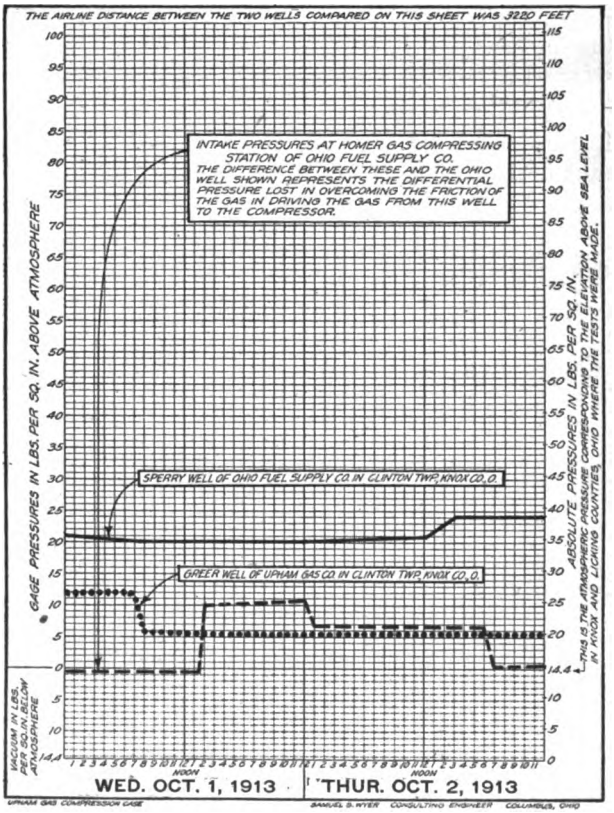

FIG. 11.—WORKING PRESSURE RELATIONS OF GAS WELL OF UPHAM GAS CO., DISCHARGING INTO LINE BY NATURAL ROCK PRESSURE WITHOUT COMPRESSORS, AND ADJACENT GAS WELL OF OHIO FUEL SUPPLY CO., DISCHARGING INTO INTAKE LINE TO HOMER GAS COMPRESSING STATION BASED ON SIMULTANEOUS RECORDING PRESSURE-GAGE RECORDS.

decrease that is inevitable with the removal of the gas from the various reservoirs.

(D) The water troubles that are experienced with certain wells in the field are due to the decrease in pressure in the local reservoir supplying the particular well, caused by the removal of the gas from such reservoir, regardless as to whether such removal was by letting the wells discharge into lines without compressors, or with compressors.

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(E) A comparison of the area of territory held to maintain and protect each producing well shows that the large companies using gas compressors maintain about 10 times more protection for their wells than the small companies not using gas compressors.

(F) Gas compressors do not induce wasteful conditions, but, on the contrary, they stimulate conservation, for the reason that, by their use, gas wells may be operated in a more economical manner.

(G) The relation of the natural or open flow to actual line flow from various wells, as determined by Pitot tube measurements, is shown in graphical form in Fig. 9.

(H) The short range of the low pressures that may be maintained in the gathering line of the gas field going to the intake side of a gas compressor is emphasized in Fig. 10.

(I) The working pressure relationship of wells discharging into intake lines to gas compressors and wells of other companies discharging into main lines not connected to gas compressors is shown in Fig. 11.

(J) 4,510 hourly pressure comparisons were made between wells of the Upham Gas Company discharging into their lines without compressors, and the wells of adjacent companies discharging into gas compressors. 90 per cent. of these showed the Upham wells to be lower than the adjacent wells, and in 26 per cent. of these the Upham wells were lower than the intakes at the gas-compressing stations drawing on the adjacent wells.

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Geology of the Ore Deposits of the Tintic Mining District

BY GUY W. CRANE,* B. A., EUREKA, UTAH

(New York Meeting, February, 1916)

I. INTRODUCTION

THE geology of the Tintic mining district, fully treated, would occupy an elaborate monograph. This less comprehensive paper is devoted primarily to the occurrence and origin of the orebodies of the district, while space is given to those phases of the general geology only which seem necessary to a proper setting of such a discussion.

The first geological report on the Tintic district to be issued was that by Tower and Smith, published in 1899, in the form of a paper in Part III of the *Nineteenth Annual Report of the U. S. Geological Survey*, and also as *Folio No.* 65, *Geologic Atlas of the United States*. About four years ago a second investigation was begun by Waldemar Lindgren and G. F. Loughlin, of the U. S. Geological Survey. With the exception of a paper on the oxidized zinc ores of the Tintic district by Mr. Loughlin, the results of their work are still unpublished. The writer's acquaintance with the district began in the fall of 1913, and has extended over a period of about two years.

1. *Location*

The Tintic district lies about 65 miles due south of Salt Lake City, and on the west central slope of the Tintic Mountains, a short range, which forms the connecting link between the Oquirrh Mountains on the north and the Canyon Range on the south—the group constituting the first of the Basin ranges to the west of the Wasatch Mountains.

The productive portion of the district embraces an area about 6 miles long and 2 miles wide, which is divided between Juab and Utah Counties. Within this area are the towns of Eureka, Mammoth, Robinson, Silver City, and Knightsville. All these points are tapped by the Denver & Rio Grande and S. P., L. A. & S. L. Railways.

2. *Topography*

As in most of the Basin ranges, topographic relief is strong, dropping rapidly from peaks of 8,100 ft. above sea level to elevations of 5,600 ft. in Tintic Valley on the west and 4,500 ft. in Goshen Valley on the east.

*Mining Geologist.

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The regularity of the range is broken by many lateral valleys, one of which has cut well back into the range, forming the low divide above the town of Eureka, through which the D. & R. G. Railway enters the district.

3. *History*

Tintic is among the oldest mining camps in Utah, being antedated only by Bingham, Rush Valley, and Little Cottonwood. Ore was first discovered in the monzonite about 1 mile east of Silver City in December, 1869, and the district was organized in the following spring. The first claim recorded was the Sunbeam, located Dec. 13, 1869, on the first discovery. The second location was made Jan. 3, 1870, on the Black Dragon, a little farther to the north. This was followed in quick succession by locations in the limestone, first on the Mammoth ledge near the middle of the district, and shortly after on the Eureka Hill ledge near its northern end.

Mining operations began at once with the construction of two mills and as many smelters during the first 12 months. Once located, the orebodies have been found to be fairly persistent, with new discoveries from month to month, and today there is good reason to believe that this record can be maintained for some time to come.

4. *Production*

During the early days of the district the production was about equally divided between the deposits in the igneous rocks at the south end of

Total Metal and Gross Values Production of the Tintic Mining District to Jan. 1, 1914.[a]

		1869 to 1879, inclusive	1880 to 1903, inclusive	1904 to 1913, inclusive	Totals to Jan. 1, 1914
Gold:	Ounces	.....	.....	831,114	.....
	Value	$475,451	$11,096,747	$17,180,602	$28,752,800
Silver:	Ounces	1,300,000	50,400,000	51,613,173	103,313,173
	Value	$1,616,130	$38,620,870	$27,343,438	$67,580,438
Copper:	Pounds	1,700,000	37,235,000	89,233,411	128,168,411
	Value	$478,852	$6,081,148	$13,473,378	$20,033,378
Lead:	Pounds	11,000,000	269,000,000	299,176,172	579,176,172
	Value	$631,146	$12,073,854	$13,869,125	$26,574,125
Zinc:	Pounds	.....	.....	7,306,281	7,306,281
	Value	.....	.....	$457,378	$457,378
Total value		$3,201,579	$67,872,619	$72,323,921	$143,398,119

[a] Compiled from statistics of G. W. Tower and G. O. Smith, U. S. Geological Survey, V. C. Heikes in reports of 1904 to 1913, inclusive, and other sources.

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the district and those in the limestone at the north end. But upon the exhaustion of the oxidized ores in the igneous rocks, due to high water-level, the balance of production was shifted to the deposits in the limestone; and since the closing of the Swansea mines at Silver City, in 1913, the entire production has been from the latter deposits.

II. Areal Geology

The rocks of the Tintic district consist of Paleozoic sediments and a variety of igneous rocks of Tertiary age.

1. *Sedimentary Rocks*

These consist mainly of quartzite and limestone having a total thickness of 12,000 ft. or more. For economic and lithologic reasons they have been subdivided in descending section as follows:

Humbug sandstone Humbug limestone	Constituting the Humbug formation of the U. S. Geological Survey.
Tetro limestone Carbonaceous shale Blue Fossiliferous limestone Chief Consolidated limestone Gemini limestone	Constituting the Godiva limestone of the U. S. Geological Survey.
White-lime-shale Centennial limestone Golden Ray limestone	Constituting the Eureka limestone of the U. S. Geological Survey.
Tintic slate Tintic quartzite	Constituting the Tintic or Robinson quartzite of the U. S. Geological Survey.

At the top of the section is the Humbug sandstone, on the east slope of the Godiva Mountain, which is 224 ft. thick, consists mainly of calcareous sandstone with a few intercalated beds of arenaceous limestone, and is not known to be ore bearing.

The Humbug limestone, 378 ft. thick, consists mainly of nearly pure, coarse-grained, gray limestone with a few intercalated beds of arenaceous limestone of a yellowish buff color. This formation is well exposed on the upper east slope of Godiva Mountain and has carried most of the ore in the Godiva-Sioux Mountain and Iron Blossom ore zones.

The Tetro limestone, 355 ft. thick, consists with little variation of hard, fine-grained, blue, cherty limestone which is not known to be ore bearing.

The Carbonaceous shale, 160 ft. thick, is essentially a thinly bedded, black, shaly limestone, containing bands of black chert, and weathering to a platy or shelly surface. It is most unpromising as an ore-bearing medium and is nowhere known to be mineralized.

The Blue Fossiliferous limestone, 542 ft. thick, consists of 300 ft. of blue, cherty limestone in beds from 1 to 3 ft. thick, resting upon 242 ft.

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of blue, highly fossiliferous limestone in thin beds separated by thin partings of impervious clay shale. This formation has been quite extensively prospected, and found to carry no orebodies of importance.

The Chief Consolidated limestone, 615 ft. thick, consists of a series of contrasted horizons which, in descending order, have the following characteristics:

80 ft. of very coarse-grained, massive, dolomitic limestone, on the whole quite favorable to mineralization.

64 ft. of fine-grained, siliceous, hard, brittle, black limestone, which is generally barren, though running the entire length of the ore zone in the Chief Consolidated mine.

38 ft. of medium-grained, fairly pure, blue limestone in thin beds, but with a few black shaly partings which appear to offer some resistance to mineralization.

31 ft. of coarse-grained, pure white to light-gray limestone which appears to be especially favorable to mineralization.

120 ft. of blue, flaky and fairly pure limestone which also appears to be especially favorable to mineralization.

72 ft. of coarse, mottled, light-gray, massive dolomitic limestone, readily susceptible to mineralization.

212 ft. of fine-grained, ashy gray, impure limestone with 14 or more intercalated thin beds of quartzitic sandstone. The horizon lies to the west of the ore zone proper, and its character is such as to discourage exploration for ore within its limits.

The Gemini limestone, 902 ft. thick, consists of 45 or more relatively thin alternating horizons of blue, gray, light-gray, and white limestone of varying texture and hardness, but all generally distinctly bedded. The formation is especially characterized by its purity, *i.e.*, lack of siliceous members, and general susceptibility to mineralization.

The White-lime-shale, 920 ft. thick, consists throughout of fine-grained, bluish gray, thinly bedded, shaly limestone with a few thin conglomeratic beds of the interformational type. It is generally extensively sheeted where folded, and weathers to a yellowish white, whence its name. It is characteristically unfavorable to mineralization, and is not known to be ore bearing.

The Centennial limestone, 798 ft. thick, consists of four relatively thick horizons of massive, dark-blue limestone interspaced by as many relatively thin horizons of thinly bedded, light-gray limestone and one 6-ft. member of thinly laminated, light-green shale. Except for the shale and two 20-ft. impure, sandy horizons, the entire formation is apparently favorable to mineralization.

The Golden Ray limestone, thickness 1,559 ft., consists of five thick horizons of heavily bedded, dark-blue to bluish gray, dolomitic limestone interspaced by five relatively thin horizons of fine-grained, light-

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gray to white, thinly laminated, impure limestone. While this formation is not known to be ore bearing, a large part of it possesses characteristics which are apparently favorable to ore deposition in other ore-bearing horizons.

The Tintic slate, thickness 358 ft., consists essentially of thinly laminated, green slate with intercalated bands of impure, gray, banded limestone near the top and thin bands of brown, quartzitic slate near the bottom. The formation is highly sheeted due to folding. Its highly siliceous and wholly barren nature does not invite exploration.

The Tintic quartzite, the lowest member in the stratigraphic series, is not fully exposed, but has an estimated thickness of from 5,000 to 7,000 ft. It is for the most part a pure, compact, fine-grained quartzite, white to pink in color. Bedding indistinct and readily confused with a pronounced sheeting due to folding. Occasional interspaced thin beds of pebble-quartz-conglomerate break the general massive nature and lend a clue to the structure.

No special effort has been given to the determination of the age divisions in the above succession. The section affords, however, no evidence of an important stratigraphical break, such as a real erosion unconformity. The thin conglomerates in the White-lime-shale are of the interformational type, and represent only temporary disturbances of the normal state of deposition during a period of shallow-sea conditions. If the chronology as determined by Tower and Smith[1] is to stand, the succession certainly represents continuous deposition from Middle Cambrian well into Pennsylvanian time.

2. *Igneous Rocks*

Packard Rhyolite.—The igneous rocks consist of rhyolite, andesite, tuff and breccia, and monzonite, all of Tertiary age.

The Packard rhyolite, locally referred to as porphyry, occurs as surface flows and intrusive dikes. It occupies a large area in the northern end of the district and, underlying the town of Eureka, extends to the north and east in a range of hills of which Packard's Peak is the most prominent. Its thickness at this point probably exceeds 1,000 ft. The rhyolite varies in color from pink and light purple to light and dark gray. In texture it ranges from glassy to highly granular, with flow banding as a common characteristic.

No metalliferous deposits are known to occur with the Packard rhyolite. Its chief economic interest lies in the numerous springs issuing from the lower slopes, from which the entire water supply of Eureka and the mines in that section of the district is obtained.

[1] Tower and Smith: Geology and Mining Industry of the Tintic District, Utah, *19th Annual Report, U. S. Geological Survey*, Part III, p. 627.

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Andesite, Tuff, and Breccia.—Large areas of andesite with associated tuff and breccia lie to the east and south of the productive portion of the Tintic district. They are, for the most part, older than the metalliferous deposits and, having no genetic relation to the latter, they need not receive further attention here.

Sunbeam Monzonite.—The monzonite is confined to the southern half of the district, where it forms the country rock over an area 4 miles in length, north and south, and 2 miles wide. It is the youngest of the igneous rocks, and occurs in the form of an intrusive stock, cutting and enveloping fragments of all the earlier formations. On the north it lies in an irregular, but nearly vertical, contact with the limestones, which it has metamorphosed through a zone varying from 300 to 500 ft. in width. On the east and south it is in irregular contact with the andesite. Its western boundary is obscured by the overlapping, alluvium-filled valleys, Ruby Hollow and Diamond Gulch, which show it to have been deeply eroded.

In color it is light to dark gray, usually with greenish brown tinge. In texture it is normally evenly granular, but varies from porphyritic on its east and west boundaries to granitic where it is in vertical contact with the limestones.

What is known as the Swansea rhyolite is probably a rhyolitic phase of the monzonite and belongs to the same period of intrusion.

The monzonite is a formation of first importance, since it has been the mineralizing source of all the important orebodies in the Tintic district.

3. *Structure*

Folding.—The sedimentary formations are folded into a simple and fairly symmetrical overturned syncline, the major axis of which strikes north and south and pitches slightly to the north. The axial plane of the fold dips about 60° to the west, resulting in vertical and often overturned dips in the beds of the west limb of the syncline, while the angles of the dip of the beds in the east limb are correspondingly low, rarely exceeding 40°. By reason of its pitch, the syncline becomes wider to the north where the trough is folded into a series of crenulations, forming a wide synclinorium. South of the Ajax fault, which extends from the Mammoth mine east to the Iron Blossom No. 3 mine, no synclinal structure is obtained. While the beds on the north of the fault are steeply inclined, conforming to the normal structure of the syncline, those on the south are relatively flat, lying with gentle dips to the east, in which direction no return dips are observable, because of a thick covering of volcanic rocks.

Faulting.—The folding of the sediments was accompanied by extensive faulting, fracturing, fissuring, and sheeting. The major faults are of the

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transverse type, striking in a nearly east-west direction across the line of the major fold. These faults show lateral displacements ranging from 500 to 3,000 ft. The direction of throw is usually to the east on the south. Connecting the major faults at various angles are numerous minor faults showing displacements ranging up to 200 ft. Rotary faulting is common, as is shown by marked disparity in the attitude of the beds along the plane of the faults. The major faults are usually accompanied by one or more parallel faults, inclosing a breccia or sheared zone. The occasional occurrence of breccia zones and slickensides parallel to the bedding planes must be interpreted as evidence of some faulting of this type. None of the faults are known to extend into the igneous rocks, which are evidently of later origin. While it is probable that most of the faulting occurred during the period of folding, it is not unlikely that some faulting and fracturing accompanied the intrusion of the monzonite, though none of this origin has been recognized.

Fractures, Fissures, and Sheeting.—Fractures and fissures occur in both the igneous and sedimentary rocks. Sheeting is confined to the latter.

In the Sunbeam monzonite and its porphyritic phase, the Swansea rhyolite, are numerous ore-bearing fissures which have a general north-south direction, but vary from as much as N. 20° W. to N. 45° E. These fissures are of the stretch type, due to shrinkage accompanying the consolidation of the igneous intrusives. For this reason they do not extend into the adjacent sedimentary rocks.

The sediments during the period of their deformation were extensively fractured and sheeted, generally along shearing planes. The fractures, which are usually steeply inclined, are most numerous in the steeply dipping beds and in the vicinity of the major faults. They trend at all angles, those in the northeast and southwest quadrants being the most abundant.

Sheeting has occurred in the quartzite, slate, and more impure shaly limestone beds, where these have been closely folded. It is best developed in the Tintic slate, and the White-lime-shale, along the western limb of the syncline. The Tintic quartzite also is extensively sheeted.

III. Geologic History

The geologic history of the district is, briefly, somewhat as follows:

During Paleozoic time the area was under water, as is evidenced by the 12,000 ft., more or less, of sediments of Paleozoic age. That conditions of sedimentation fluctuated considerably between shallow and deep waters, is confirmed by the alternation of thinly bedded shaly limestone, and shale with relatively pure massive limestones; but there is no evidence that, during this age, any portion of the area was elevated above sea level even for a brief interval.

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Early in Mesozoic time extensive earth movements began, which resulted in the elevation and deformation of the Paleozoic sediments into virtually their present highly disturbed condition. Throughout this period of folding and faulting, erosive agents were active in carving the distorted strata into mountain ranges closely approaching the present forms. While folding had probably ceased some time before, the erosion continued right up to the Tertiary.

During Tertiary time, the greater portion of the area was buried beneath extensive flows of rhyolite and andesite, which were interspaced by short periods of erosion. Where the rhyolite flows overlapped the sediments, considerable quantities of iron oxide and ferruginous jasper were formed. These are described under the head of limestone-igneous contact deposits.

Following the extrusion of the Packard rhyolite and andesite, with at least a short period of erosion intervening, came the intrusion of the Sunbeam monzonite and its quartz-porphyry phase, the Swansea rhyolite.

This event brought about the second period of ore deposition, which resulted in the formation of the replacement deposits in limestone and was accompanied or soon followed by the deposition of the fissure-vein ores in the Sunbeam monzonite and Swansea rhyolite. Since the last of the volcanic eruptions and the period of ore deposition, there has been uninterrupted erosion, which has cut deeply into the accumulations of volcanic materials, exposing extensive areas of the sediments and, at a few points, the inclosed orebodies.

The products of this last period of erosion form the alluvium in the valleys and the talus on the upper slopes.

IV. The Ore Deposits

There are three types of ore deposits in the Tintic district: (1) limestone-igneous contact deposits; (2) deposits in igneous rocks; and (3) limestone replacement deposits.

1. *Limestone-Igneous Contact Deposits*

At numerous points along the contact of the limestone and the igneous rocks, particularly the Packard rhyolite, are thin deposits of iron and manganese oxide associated with more or less red and greenish yellow jasperoid and clay-ironstone. The jasperoid has been observed in contact with the limestone of which it is a replacement. The iron-manganese deposits occur both at the contact and filling cavities in the limestone near the contact.

Deposits of this type are widely distributed over the district, particularly its northern end, where, because of the resemblance of the

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associated jasperoids to certain phases of the quartz in the productive limestone replacement deposits, they have received considerable attention from prospectors.

Locally small quantities of silver and gold have been reported; but, except for the iron ores which have been mined intermittently for fluxing purposes, these deposits have no economic importance.

2. *Deposits in Igneous Rocks*

The ore deposits in the igneous rocks are confined to the southern half of the district, where they occur as veins in well-defined stretch fissures in the Sunbeam monzonite and Swansea rhyolite. The veins are as a rule nearly vertical with frequent croppings at the surface. They have general north-south courses but vary from N. 20° W. to N. 45° E. One, the Sunbeam, has been worked through a length of 2,000 ft. on the strike, but most of them pinched out at a distance of a few hundred feet.

As shown by the old workings, the veins vary in width from a few inches to 10 ft., usually pinching with depth. The average width is about 2 ft. The ore in its unaltered state consisted of silver-lead sulphides and sulpharsenides, occurring in bands and lenses in a gangue of quartz and pyrite. These materials constituted about three-quarters of the mass of the vein. Considerable copper occurred in some of the southern mines, while those in the north have produced lead and silver almost exclusively. Gold occurred sparingly in but few of the deposits. Ground-water level was usually encountered at a depth of from 200 to 700 ft., depending upon the proximity of the sediments. Above water level, the ores assumed the usual oxidized character, with some secondary enrichment. Below ground-water level there was marked decrease in the value and size of the orebodies. This and water troubles usually caused an early cessation of mining, and today all these properties, with one or two exceptions, are indefinitely abandoned. Since the closing of the Swansea mines in 1913 there has been no noteworthy production from deposits of this type.

3. *Replacements in Limestone*

The orebodies of this type are confined to the northern half of the Tintic district, and include all the now productive mines. Mining operations have developed four nearly parallel ore-bearing zones, within which the orebodies appear to be virtually continuous, though at intervals the connections may be mere seams of quartz. These zones have a general north-south direction and may be defined as follows:

1. The Eureka zone, including the Centennial Eureka mine on the south and extending through the Eureka Hill, Bullion Beck, and Gemini mines to the Ridge and Valley mine on the north.

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2. The Mammoth zone, beginning at the Black Jack mine on the south and extending through the Lower Mammoth, Phœnix, Gold Chain, Mammoth, Grand Central, Victoria, Eagle, and Blue Bell mines to the Chief Consolidated mine on the north.

3. The Godiva-Sioux Mountain zone, beginning at the North Star mine on the south and extending through the Red Rose, Carisa, Northern Spy, Utah, Uncle Sam, Yankee, May Day, and Apex mines to the Godiva mine on the north.

4. The Iron Blossom zone, beginning at the Dragon Consolidated mine on the south and extending through the Black Dragon, Governor, Iron Blossom Nos. 1 and 3, Sioux Consolidated, Colorado Nos. 1 and 2, and Beck Tunnel No. 1 mines to the Beck Tunnel No. 2 mine on the north.

As might be inferred, these four zones include all the mines that have produced this type of ore. The Eureka zone is the shortest, and at its northern end is divided into two or more channels. At its southern end, it appears to connect with the Mammoth zone, through the Grand Central mine, in which case it is a branch of the Mammoth zone. These two zones lie on the west limb of the syncline, where the beds are nearly vertical and their orebodies have great vertical range.

The other zones are well defined throughout their length of about 2 miles and apparently have no interconnections. They lie near the trough of the syncline and their orebodies are correspondingly shallow.

Occurrence of the Ore.—Like replacement deposits in general, those of Tintic are very irregular in form, but in both shape and size they are governed within certain limits by the associated structures. This explains the dissimilarity in occurrence of the orebodies in the different zones.

In the Eureka and Mammoth zones, which are in the nearly vertical beds of the west limb of the syncline, ore has been mined from the surface to the depth of 1,800 ft.; the orebodies as a rule conforming very closely to the bedding in both strike and dip.

If any one form of orebody can be said to be typical of these zones, it is that of an irregular sheet, pinching and swelling along the line of the beds, and feathering upward into chimneys or pipes.

In an earlier report on the Tintic district[2] reference is frequently made to the north-south fissures, which, it is contended, have governed the course of the ore channels. The writer's investigations have not confirmed the common presence of such north-south fissures within the ore zones, but, on the contrary, have gone to show that practically all the major faulting and fissuring in the district is of the nearly east-west or transverse type.

[2] Tower and Smith: Geology and Mining Industry of the Tintic District, Utah. *19th Annual Report, U. S. Geological Survey*, Part III.

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Where the ore channel is crossed by faults or zones of fracture, there is usually an enlargement of the orebody along the line of intersection of the faulting and the bedding. This gives rise to another very common type of orebody, characteristic of the western zones, where it is usually the most largely productive form.

At the intersection of the ore channel by a fault, it is not an uncommon occurrence to find that the line of mineralization turns from the plane of the beds and follows the fault for distances up to 100 ft. or more, before again taking the usual course along the beds. Occurrences of this kind probably also account for the occasional division of a single ore channel into two or more parallel ore channels.

In the Godiva-Sioux Mountain zone, the known orebodies have smaller vertical extent but greater lateral variation than do those in either the Eureka or Mammoth zones. This is probably due to the fact that the Godiva-Sioux Mountain zone lies nearer the trough of the syncline where the beds dip at relatively lower angles, seldom exceeding 60° and sometimes as low as 15°.

In the Iron Blossom zone, particularly in its northern half, we find still greater restriction in both vertical and lateral extent of the ore than in any of the other ore zones. This, again, can be attributed to the controlling influence of the structure, since this portion of the zone takes a course directly in the trough of the syncline. The south half of the zone traverses beds dipping at angles up to 30°, and here the orebodies resemble more closely in form those of the Godiva-Sioux Mountain zone.

Influence of Texture and Composition.—Next in importance to structural control is that of the texture and composition of the limestone. Other things being equal, the coarser-grained, softer, and less siliceous limestones are more susceptible to mineralization than are the fine-grained, more siliceous and harder ones, while the associated shales, sandstones, slates, and quartzite are uniformly barren. This principle is well exemplified in all the Tintic replacement deposits, as well as in the replacement deposits of other districts.

Among the several formations described above, there are only four which are known to be ore bearing. These are the Centennial, Gemini, Chief Consolidated, and Humbug limestones. In each case the ore-bearing formation contains several horizons of coarse, soft, and relatively pure limestone, while the remaining formations are more generally fine-grained, hard and siliceous, in that they carry varying proportions of shale and sand.

The influence of magnesia in the limestone is not clearly shown One dolomitic horizon in the Chief Consolidated formation is quite generally mineralized, while another is only sparingly so. The dolomites are usually coarse grained—a favorable factor, which may offset the disadvantages of difference of solubility.

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Zone of Oxidation.—By reason of the folded and broken state of the limestone beds, the ground-water level is extremely low (about 4,765 ft. above sea level), in the sedimentary rocks. This puts it at depths beneath the surface of from 2,300 ft. in the higher properties to 1,500 ft. in those having shafts at relatively lower positions. This condition has led to more or less complete oxidation of the original sulphide ores, with some downward segregation of the ore minerals, and some enrichment due to shrinkage in volume.

V. The Ore

Very little ore has been mined below ground-water level; but developments have demonstrated its continuity into that zone, and something as to its original sulphide character. Above the water table the ore is largely oxidized.

1. *Physical Character*

The oxidized ore is usually massive and granular and quite without structure. A distinct banding, observed locally, is attributed by some to the preservation of an original banding in the replaced limestone. Some soft ocherous material, which usually caps the orebodies, is thinly laminated or stratified—the effect of transportation and redeposition during the process of oxidation. Material of this type is usually found capping the larger bodies of oxidized ore and is encountered more particularly in the higher levels of the oxidized zone.

2. *Mineral Composition*

The ore minerals contain gold, silver, lead, copper, and zinc, in a gangue composed mainly of quartz and some pyrite in the unaltered ore, but with important quantities of calcite and iron and manganese oxides, and smaller amounts of barite and dolomite in the oxidized ore.

Gold is seldom visible, but occurs locally in the native state. Silver occurs native, and where unoxidized as the sulphide (argentite) and the antimonial sulphide (stephanite); but the original compounds are usually altered to the chloride (cerargyrite, or horn silver). Lead occurs originally as the sulphide (galena), but is largely altered to the carbonate (cerussite), with smaller amounts of the sulphate (anglesite), and one or two rarer forms. Copper occurs originally as the sulpharsenide (enargite), but is usually altered to the hydrous silicate (chrysocolla) and the hydrous carbonates (malachite and azurite). Some native copper has been observed which may be in part original. Zinc occurs originally as the sulphide (sphalerite), but is usually altered to the carbonate (smithsonite) or the hydrous silicate (calamine), with small amounts of one or two rarer forms.

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3. *Distribution of the Ore Minerals*

From the earliest days an important variation in the north and south distribution of the ore minerals in the several ore zones has been noted. It appears that gold-copper ores predominate at the south end of the zones, gradually giving place to increasing proportions of silver-lead ores to the north, where the latter minerals predominate. As a result, the mines in the vicinity of Mammoth have been chiefly producers of copper and gold, while those in the vicinity of Eureka are chiefly producers of silver and lead. Zinc is found associated with the ore of both ends of the district, probably more frequently in the mines at the north. Vertically the ore zones show no such striking variation in mineral content; but generally the good gold stopes are in the upper levels.

The vertical extent of an ore zone is governed primarily by the accompanying structures. The more nearly vertical are the inclosing beds the greater the vertical range of the orebody. The presence of faults and fissures also favors greater vertical range, irrespective of the angle of the bedding. Their influence is greatest, of course, where the bedding is nearly vertical. The large orebody in the south end of the Chief Consolidated mine, which has a vertical range of 1,000 ft. or more, rakes along the inclined line of intersection of vertical beds and a large cross break.

4. *Origin of the Ore*

The occurrence of the Tintic deposits both within the monzonite and in the immediately adjacent limestones; the character of the mineralization, which is now generally considered to be due to the action of hot solutions; the progressive change in the mineral composition of the ore with increased distance from the monzonite—all point to this rock as the prime factor in the mineralization. In this respect, the deposits of Tintic are like the many others in the southwestern part of the United States, and particularly in western Utah, which are as a rule closely connected in origin with igneous (commonly monzonite) intrusions.

The localization of the deposits within the monzonite was determined by the formation of stretch fissures within the congealing magma, which furnished channels of circulation for the hot solutions that were given off during the period of cooling. These hot solutions chloritized the ferromagnesian minerals, sericitized the feldspars of the wall rocks, and filled the fissures with an aggregate of quartz, pyrite, galena, sphalerite, barite, and some copper, silver, and gold minerals, probably because of a decrease in pressure and temperature as the solutions approached the surface. Alteration above ground-water level has changed these minerals to their oxidized equivalents, with some secondary enrichment.

The localization of the deposits in the limestone was controlled in

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part by the attitude and character of the beds and in part by the presence of fissures, some of which were probably produced during the period of monzonite intrusion.

Following the fissures caused by the intrusion, the hot mineral-bearing solutions found their way through the fractured and metamorphosed zone into the unaltered limestone, where they naturally took the easiest avenue afforded. This, in the absence of open faults and fissures, was the plane of the bedding.

Where the dip of the beds approached the vertical, the solutions found channels at various elevations, ascending or descending as conditions allowed, but laterally confined to relatively narrow limits. Flat-lying and low-pitching beds, on the other hand, have tended to confine the solutions to a more nearly horizontal course, at the same time permitting greater lateral shift.

At the intersection of faults, the solutions were allowed to spread out both vertically and laterally, producing orebodies of large size and great vertical range.

The process of mineralization has been primarily a replacement of the limestone with quartz and varying proportions of copper, lead, zinc, and silver sulphides, presumably due to decreasing temperature and pressure in the solutions as they run their northerly course.

Soft, coarse-grained, nearly pure limestones have been much more readily replaced by the solutions than the harder, fine grained, and more siliceous beds. Certain horizons of the latter type, even where directly in the course of the mineralized zone, are found to be uniformly barren.

VI. Prospects for the Future

Of the four zones so far developed, not one has been traced to a logically demonstrated terminus; and it is by no means established that similar and equally productive zones do not exist, still undiscovered, in the district.

Since the inception of mining, following the first discovery in 1869, the trend of production in the district has been steadily upward. Great mines have come and gone; but there have been always new ones to take their places. The greatest production in the district's history was attained but two years ago, and that of the coming twelve months promises to exceed it. In view of these facts, and of the tendency to better and more improved methods of mining as our knowledge of the deposits is extended, the conclusion is inevitable that the Tintic district has still to experience the most prosperous period in its history.

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The Disseminated Copper Ores of Bingham Canyon, Utah

BY J. J. BEESON, STANFORD UNIVERSITY, CAL.

(New York Meeting, February, 1916)

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I. Primary Mineralization

1. *Introduction*

a. Scope of Work.—The first part of this paper deals with the country rock, its pegmatitic, pneumatolytic, and hydrothermal alterations; also, the effect of magnetite on the deposition of the primary chalcopyrite and pyrite.

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The second part of the paper includes a detailed account of the processes of secondary enrichment which have been effective in producing the disseminated ores. While both parts present new features, it is hoped that the discussion on secondary chalcopyrite and bornite may be of interest to those who are engaged in the study of secondary processes in other districts.

The excellent work,[1] Economic Geology of the Bingham Mining District, by J. M. Boutwell, was published in 1905. Since that time the extensive development work—surface cuts and underground drifting—of the Utah Copper Co. has made possible the collecting of samples for more detailed microscopic study. The writer has taken advantage of this fact and presents the following work, not as a criticism, but as a continuation of the work which was begun by Mr. Boutwell.

b. Acknowledgments.—The present study was undertaken at the suggestion of Dr. A. F. Rogers; to him, and to Prof. C. F. Tolman and Prof. S. W. Young, the writer wishes to express his thanks for criticism in the preparation of this paper.

The writer also expresses his thanks to D. C. Jackling, Vice-President and Managing Director of the Utah Copper Co., for permission to pub lish this paper, and to William Spencer, D. W. Wadleigh, T. S. Carnahan, J. A. Caulfield, Percy Dayer, and Edward Norsefall, for the courtesies extended to him while a co-worker in the mines.

c. The Bingham Mining District.—The Bingham mining district is situated in the Oquirrh Mountains, 18 miles southwest of Salt Lake City, Utah. The productive region is limited to an area of about 15 square miles; but the largest producing mines are included in an area of about 2 square miles in the southwestern end of the district.

d. General Geology.—Since the general geology, maps, sections, and detailed descriptions of the different types of ore deposits appear in Mr. Boutwell's report, the writer will only summarize briefly the general geology, emphasizing certain important points mentioned by Mr. Boutwell.

The sedimentary section exposed in this district embraces several thousand feet of massive quartzite, thin interlaced limestones, and calcareous shales of the Upper Carboniferous (Pennsylvanian) age. These sediments have been cut by an intrusive quartz-monzonite which forms connected irregular dikes and masses, in a strip about 1 mile wide and 4 miles long, extending across the southern corner of the district in a northeast-southwest direction.

Bordering the quartz-monzonite are contact deposits, replacement

[1] J. M. Boutwell: Economic Geology of the Bingham Mining District, Utah. *Professional Paper No. 38, U. S. Geological Survey* (1905).

J. M. Boutwell: Genesis of the Ore-Deposits at Bingham, Utah. *Trans.*, xxxvi, 541 to 580 (1905).

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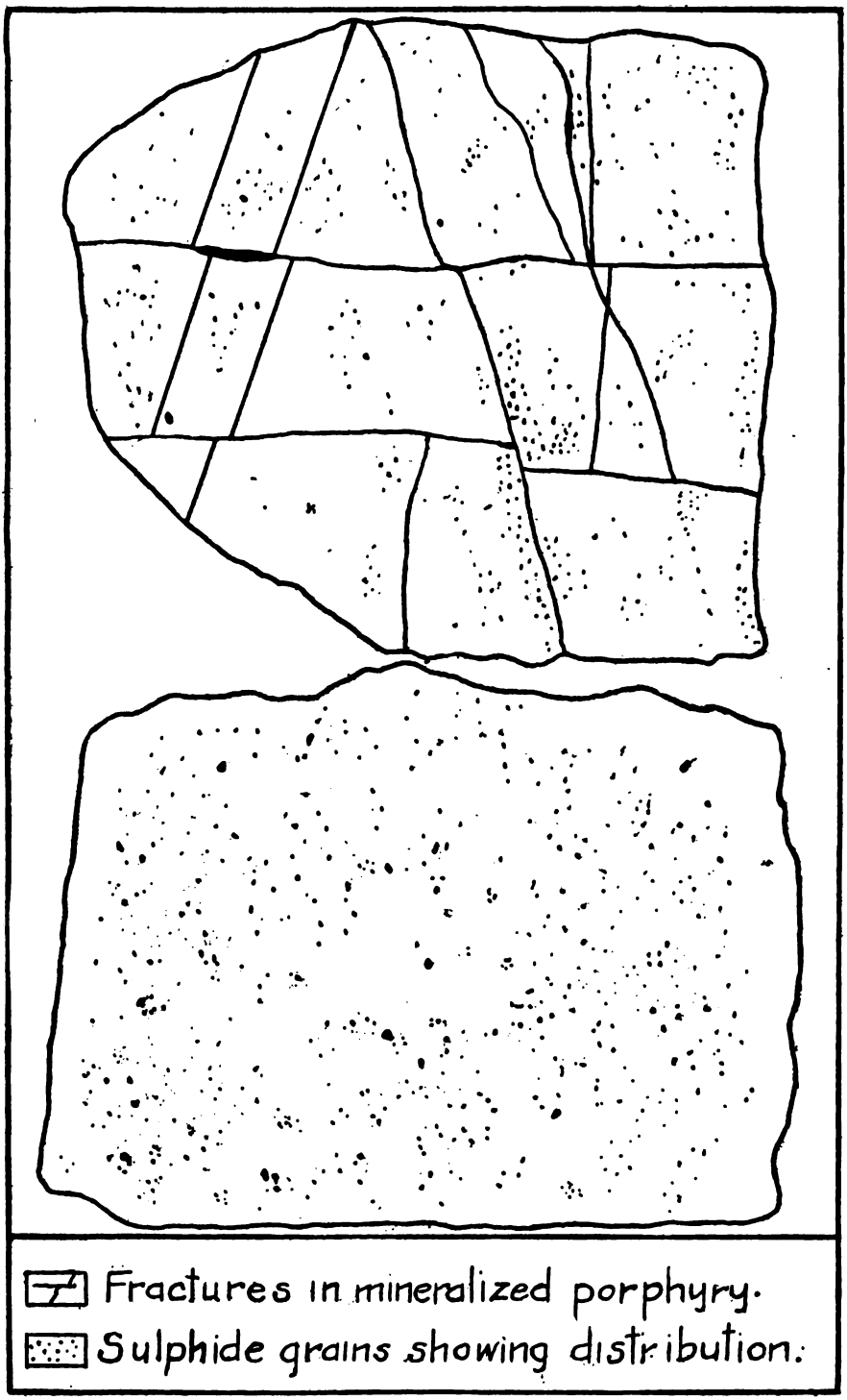

PLATE I

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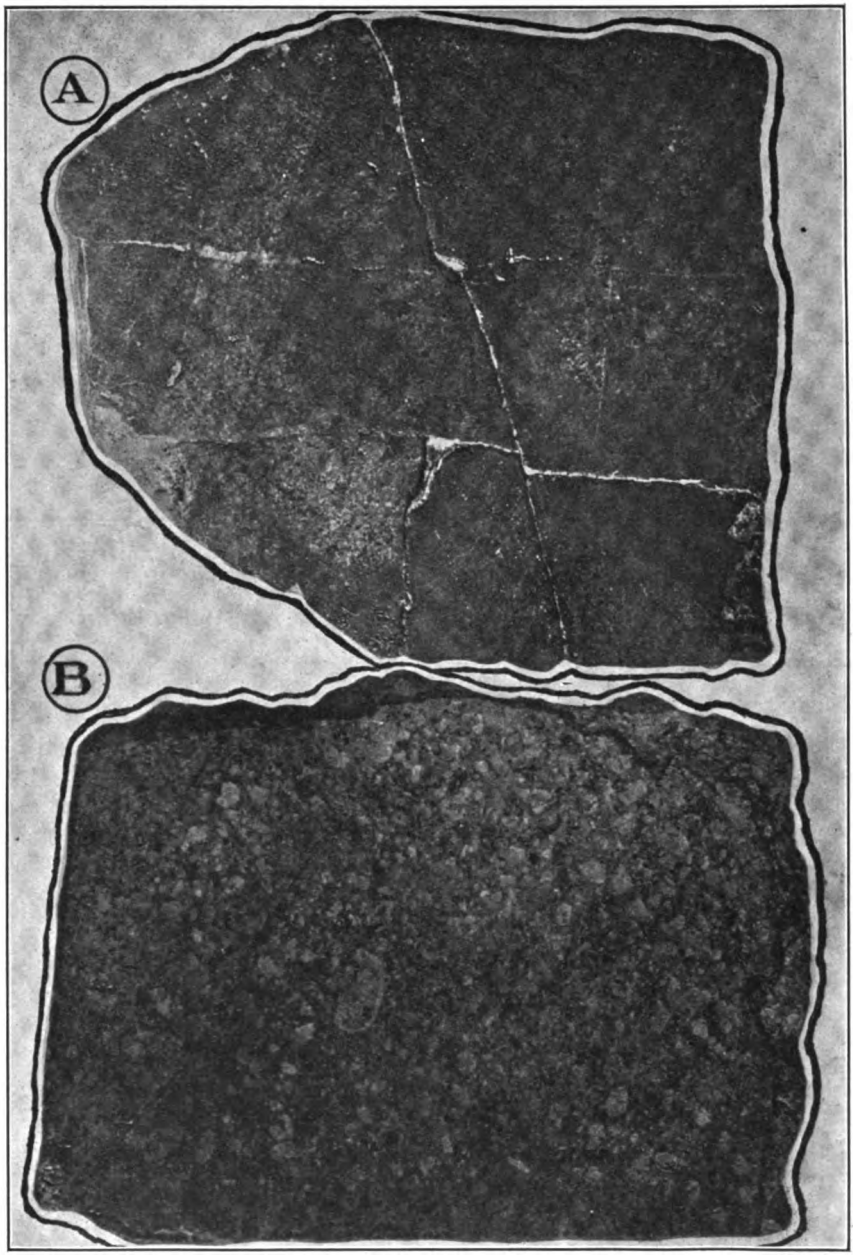

Mineralized Quartz-Monzonite from Utah Copper Underground Mine. Natural size.
PLATE I

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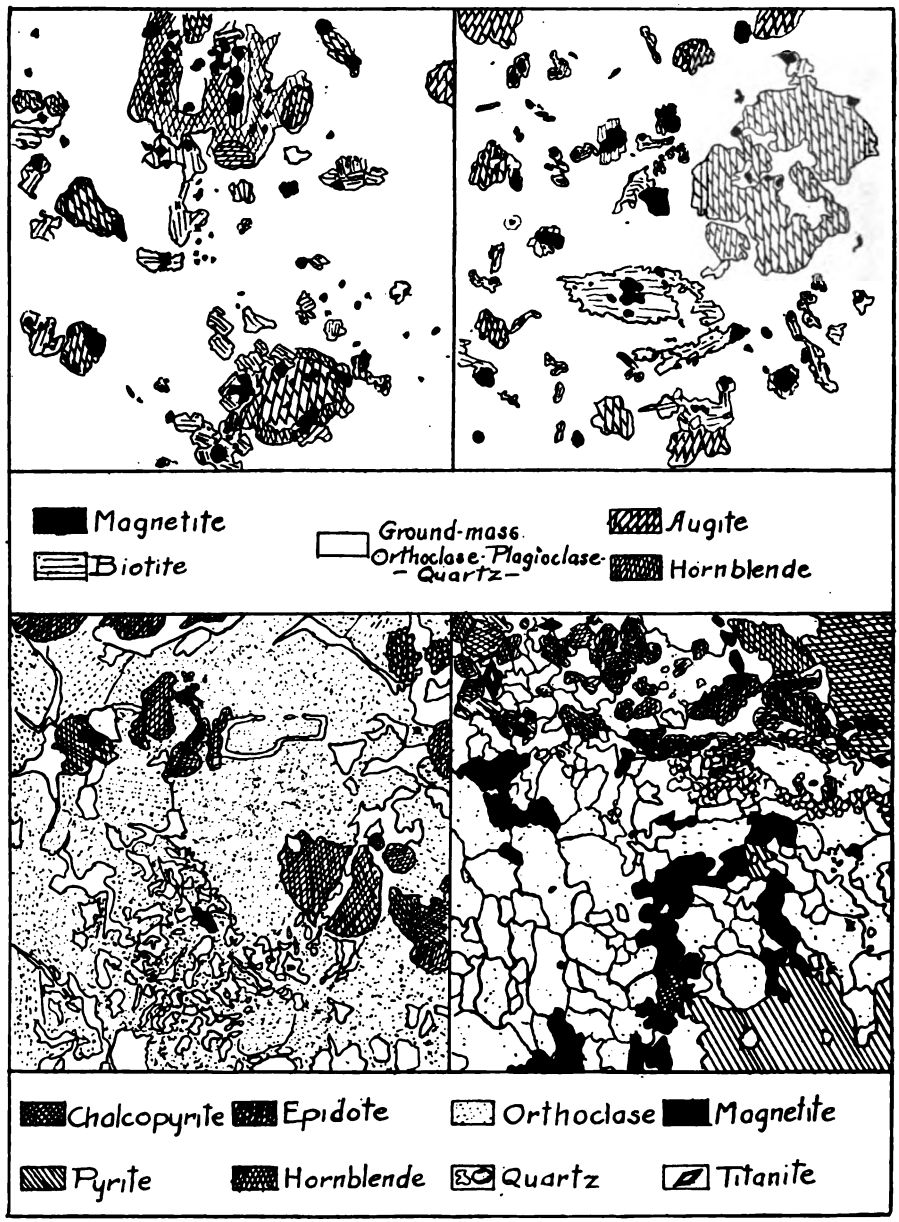

PLATE II

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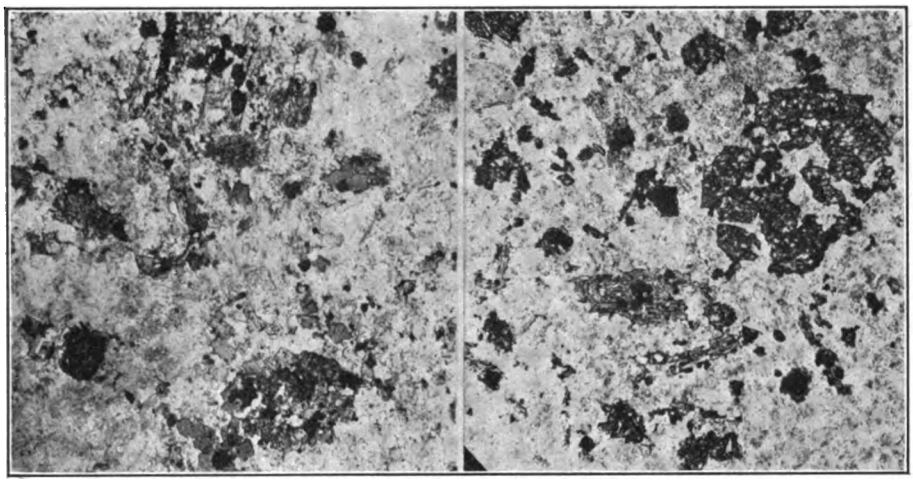
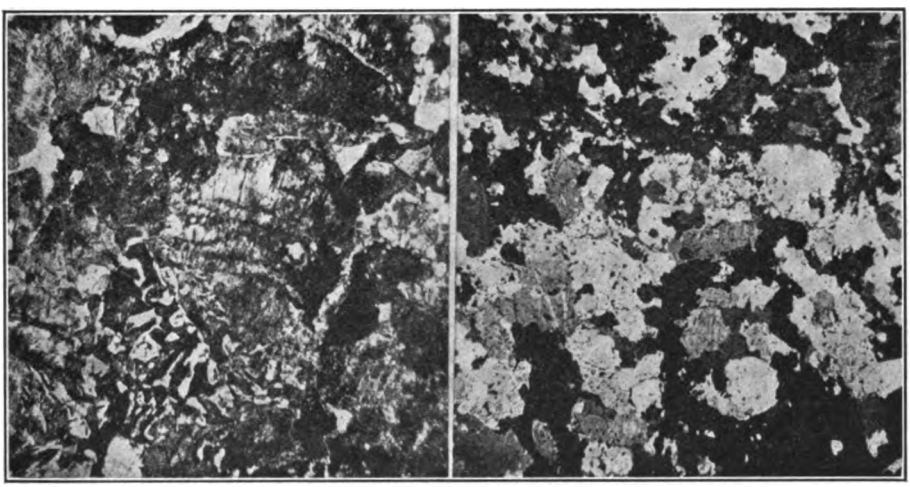


A.—Quartz-Monzonite from near Silver Shield Mine. × 10 diameters. Ordinary light.

B.—Quartz-Monzonite from near Highland Boy Mine. × 26 diameters. Ordinary light.



C.—Pegmatite Vein from Commercial Mine. × 6 diameters. Parallel nicols.

D.—High-Temperature Quartz Vein from Commercial Mine. × 27 diameters. Crossed nicols.

PLATE II

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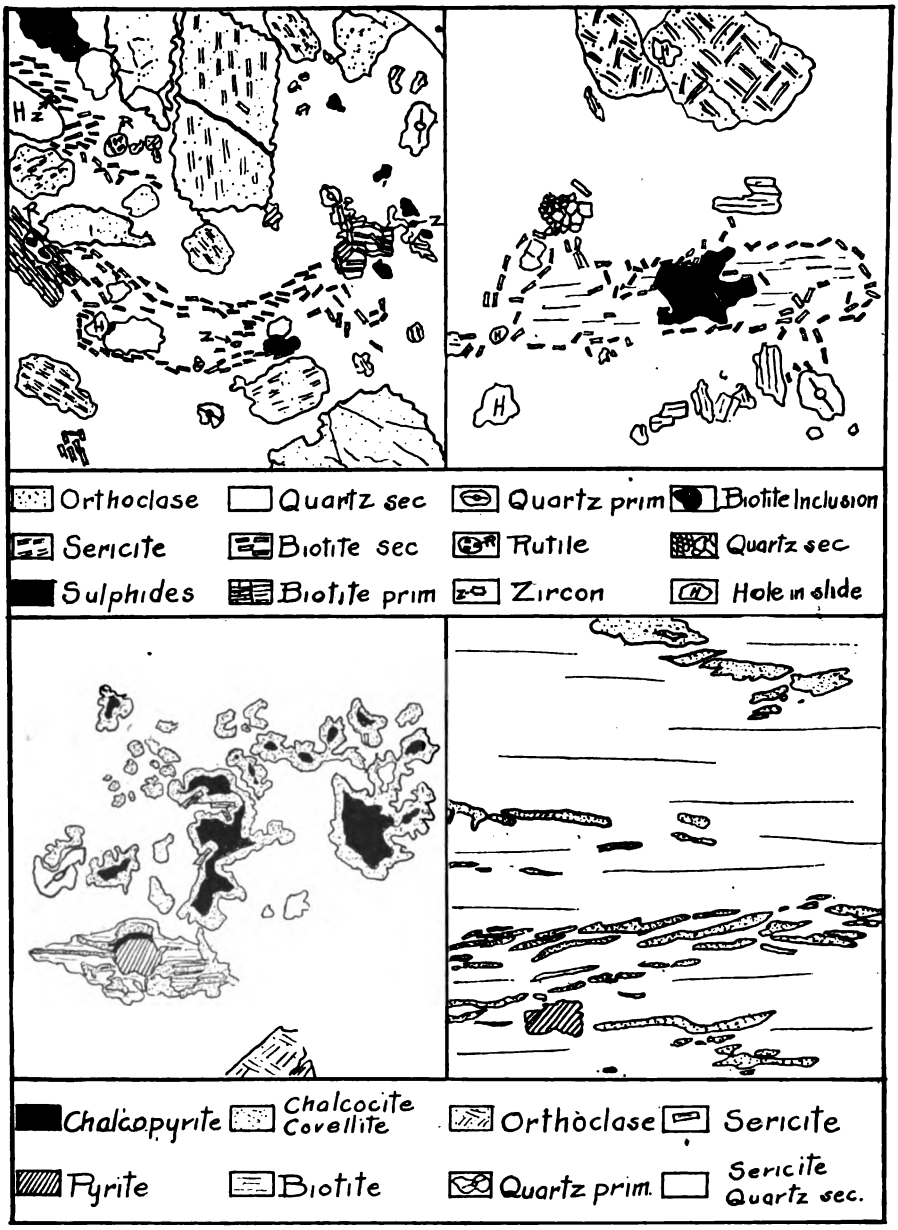

PLATE III

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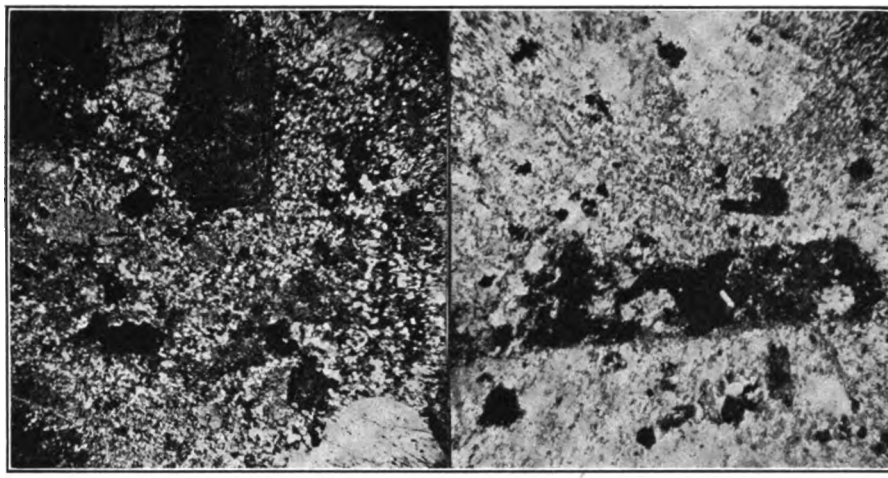
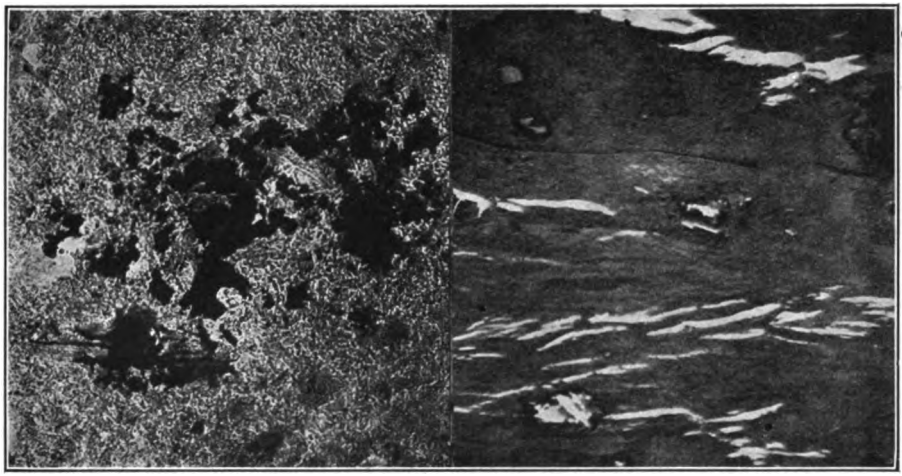


A.—Metasomatic Replacement of Feld spar Groundmass by Secondary Quartz. Mineralized Quartz-Monzonite. × 8 diameters thin section. Crossed nicols.

B.—Chalcopyrite in Altered Crystal of Biotite. From hand specimen *B*, Plate I. × 12 diameters thin section. Crossed nicols.



C.—Pyrite Grains in Biotite, Chalcocite, Covellite, along Cleavage Planes. Mineralized Quartz-Monzonite. × 51 diameters thin section. Ordinary light.

D.—Enlargement from Same Specimen as *C*, showing Chalcocite as Gashes in Biotite Crystal. × 100 diameters. Polished section.

PLATE III

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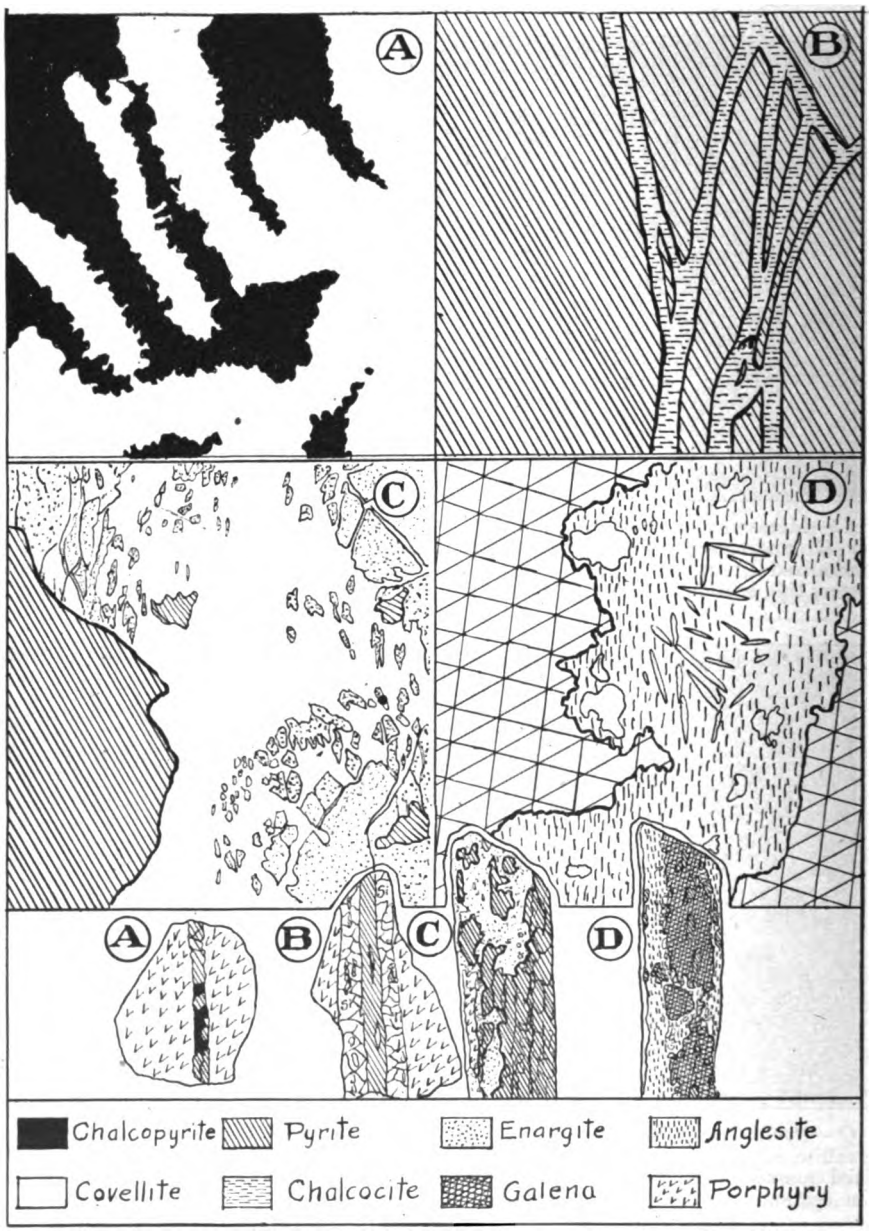

PLATE IV

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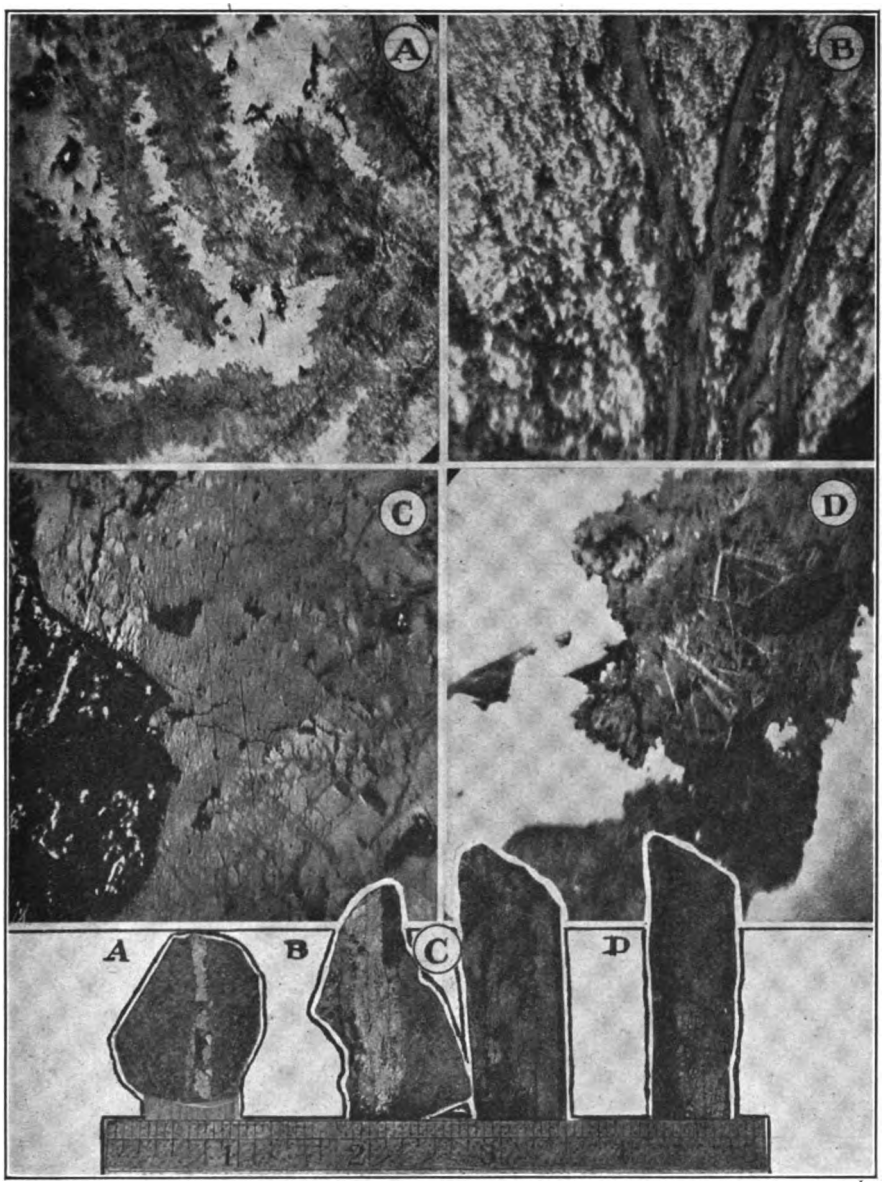

A.—Covellite Replacing Chalcopyrite. × 96 diameters.

B.—Chalcocite Replacing Pyrite. × 241 diameters.

C.—Covellite Replacing Enargite. × 72 diameters.

D.—Covellite Replacing Galena. × 74 diameters.

PLATE IV

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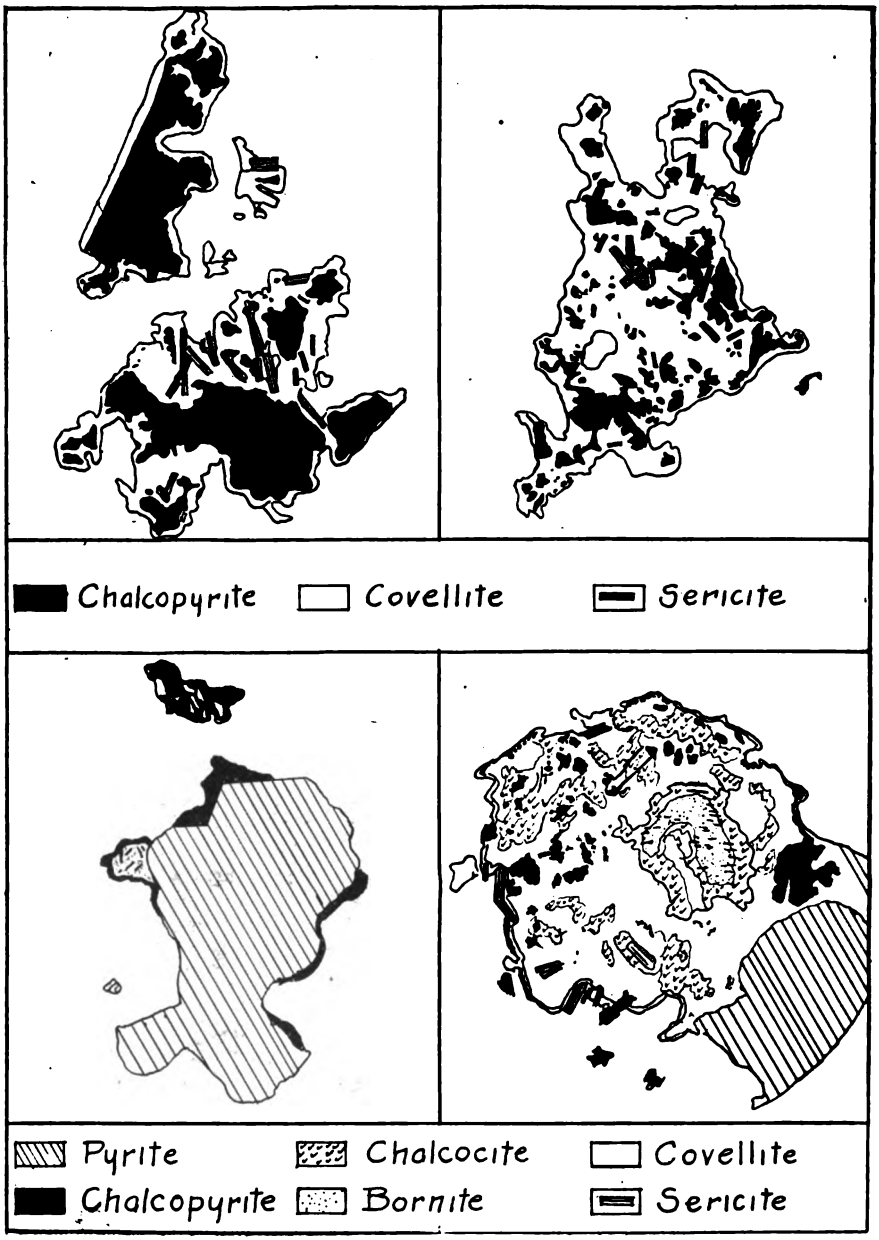

PLATE V

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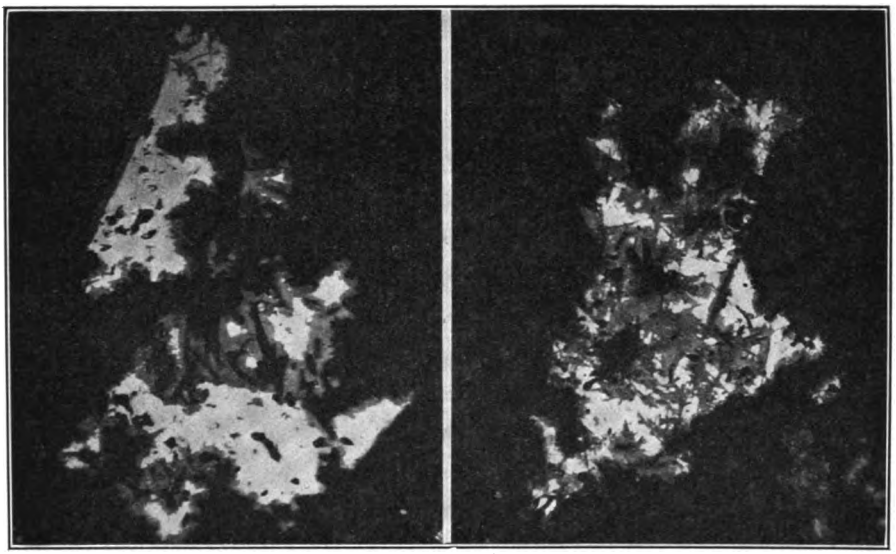
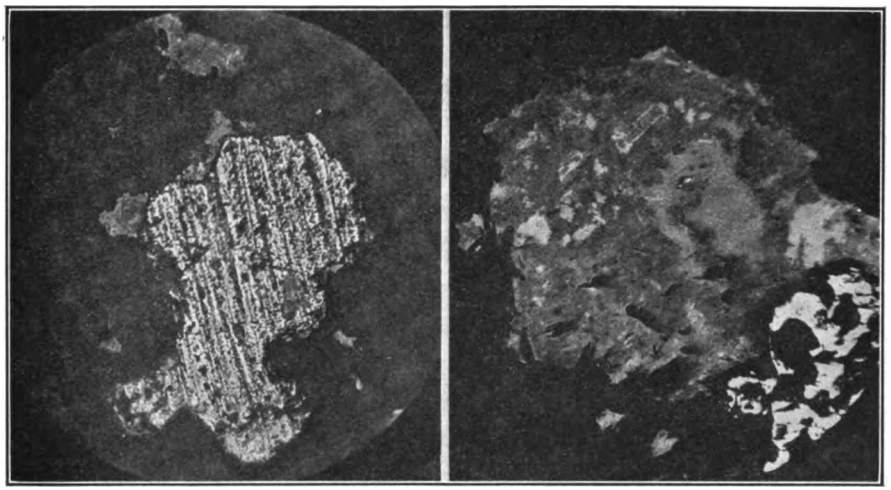


A.—Covellite Replacing Chalcopyrite. Covellite Ores, Underground Mine. × 129 diameters. Polished section.

B.—Covellite Replacing Chalcopyrite. Covellite Ores, Underground Mine. × 103 diameters. Polished section.



C.—Pyrite, Chalcopyrite, Bornite. Covellite Ores, Underground Mine. × 96 diameters. Polished section.

D.—Pyrite, Chalcopyrite, Bornite, Chalcocite, Covellite. Covellite Ores. × 55 diameters. Polished section.

PLATE V

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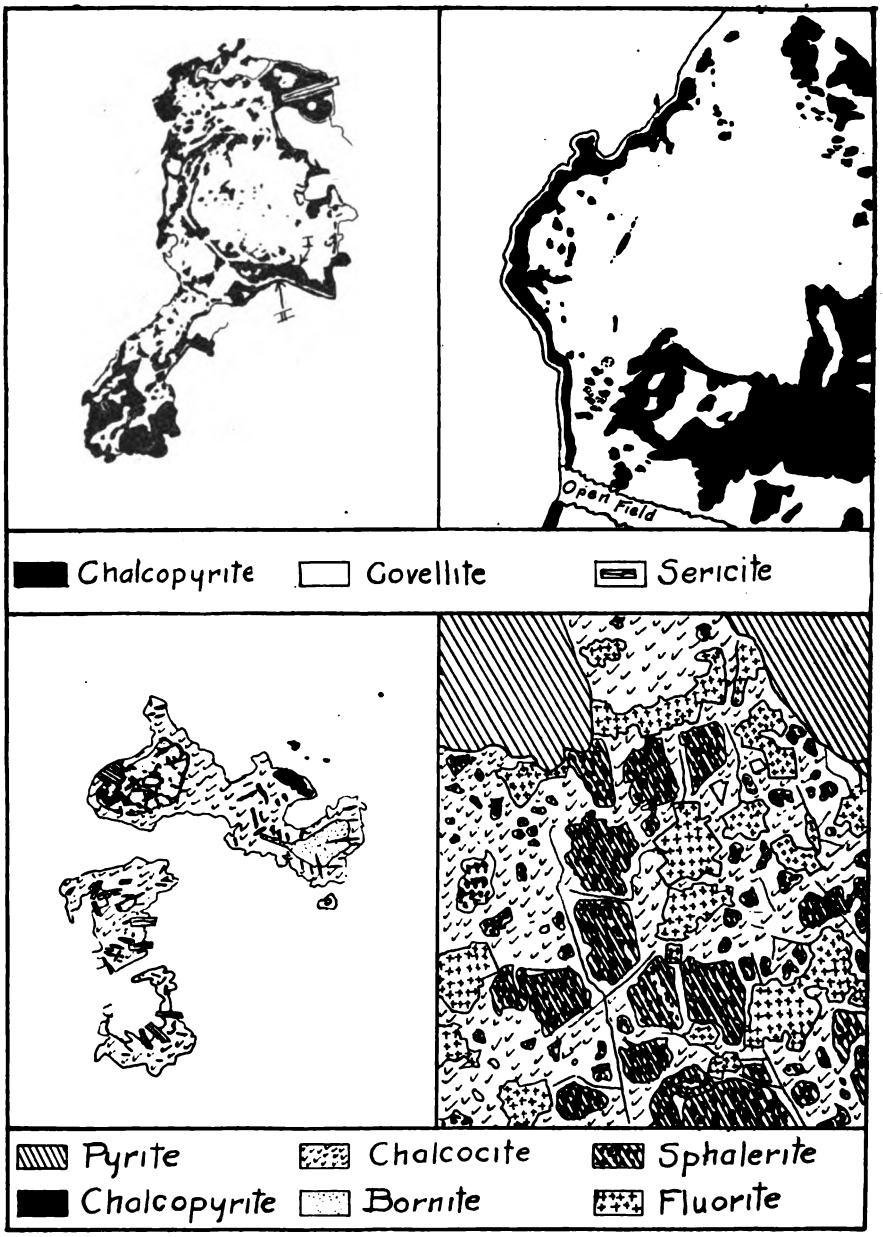

PLATE VI

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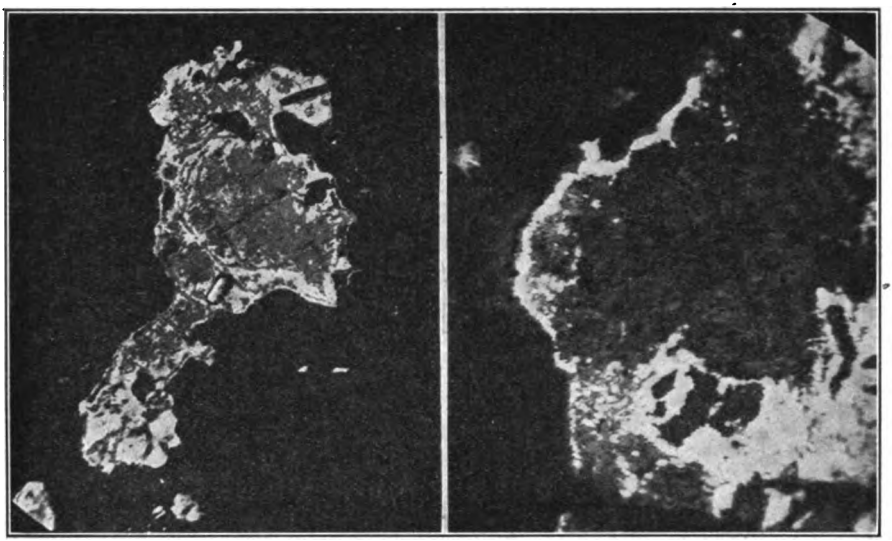
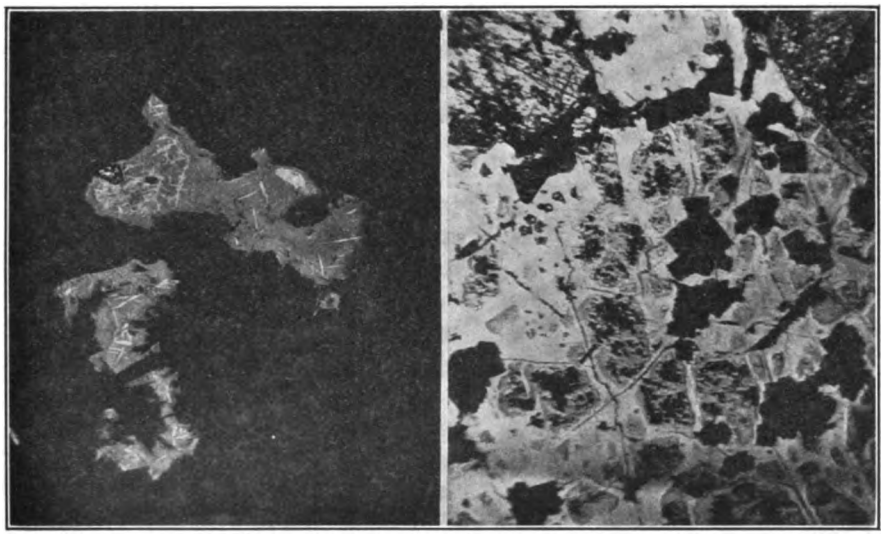


A.—Chalcopyrite Replacing Covellite. Covellite Ores. × 96 diameters. Polished section.

B.—Chalcopyrite Replacing Covellite. Covellite Ores. × 212 diameters. Polished section



C.—Chalcocite Replacing Bornite. Covellite Ores. × 55 diameters. Polished section.

D.—Chalcocite Replacing Sphalerite. Covellite Ores. × 96 diameters. Polished section.

PLATE VI

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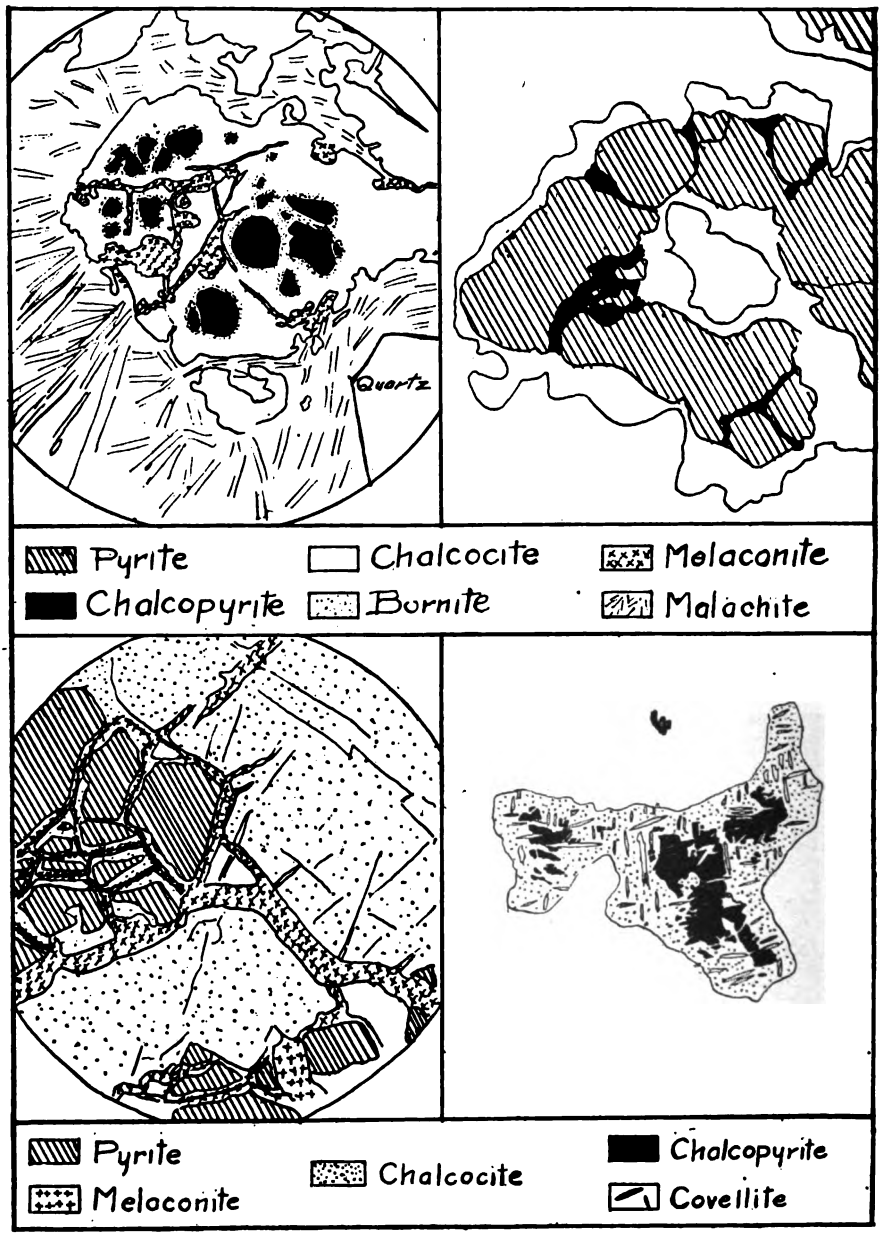

PLATE VII

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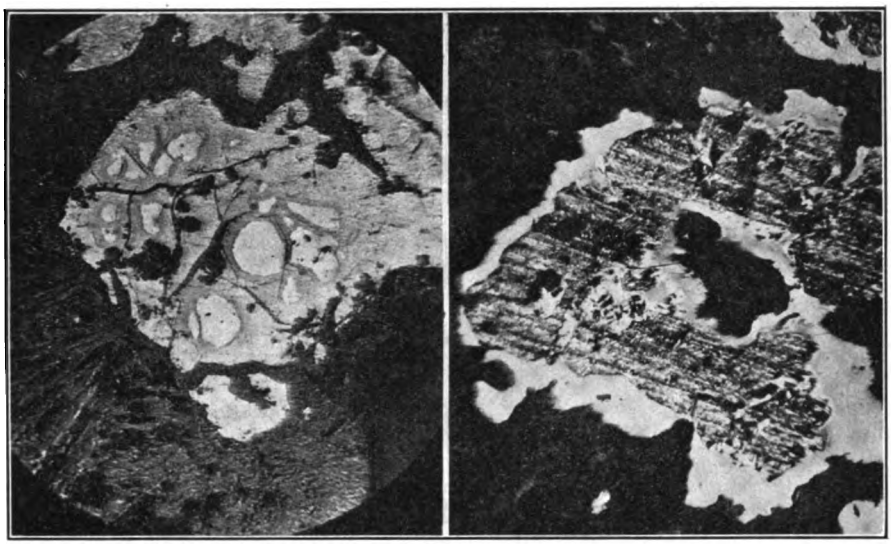
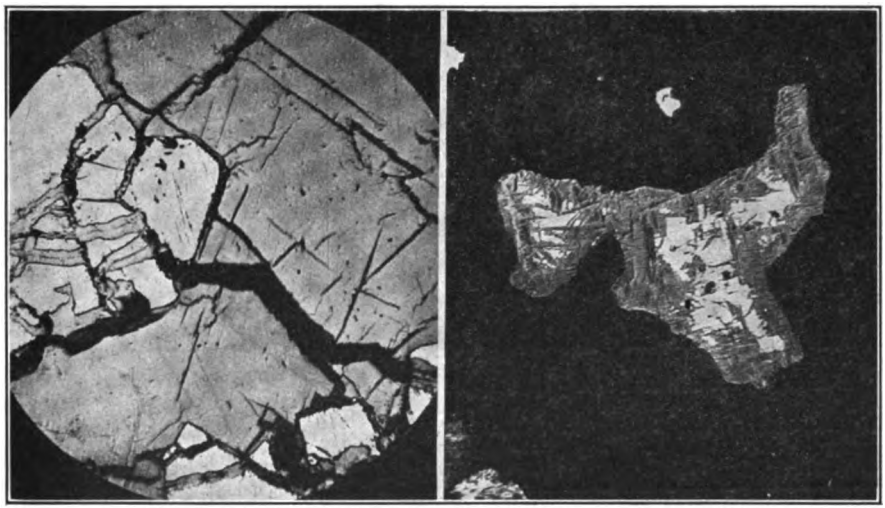


A.—Chalcocite Replacing Chalcopyrite with Bornite as an Intermediate Product. Chalcocite Replaced by Melaconite. × 58 diameters. Polished section.

B.—Chalcocite Replacing Pyrite and Chalcopyrite. Chalcocite Ores. × 116 diameters. Polished section.



C.—Chalcocite Replacing Pyrite. Melaconite Replacing Chalcocite. Chalcocite Ores. × 55 diameters. Polished section.

D.—Covellite Replacing Chalcopyrite. Chalcocite Replacing Covellite and Chalcopyrite. ×64 diameters. Polished section.

PLATE VII

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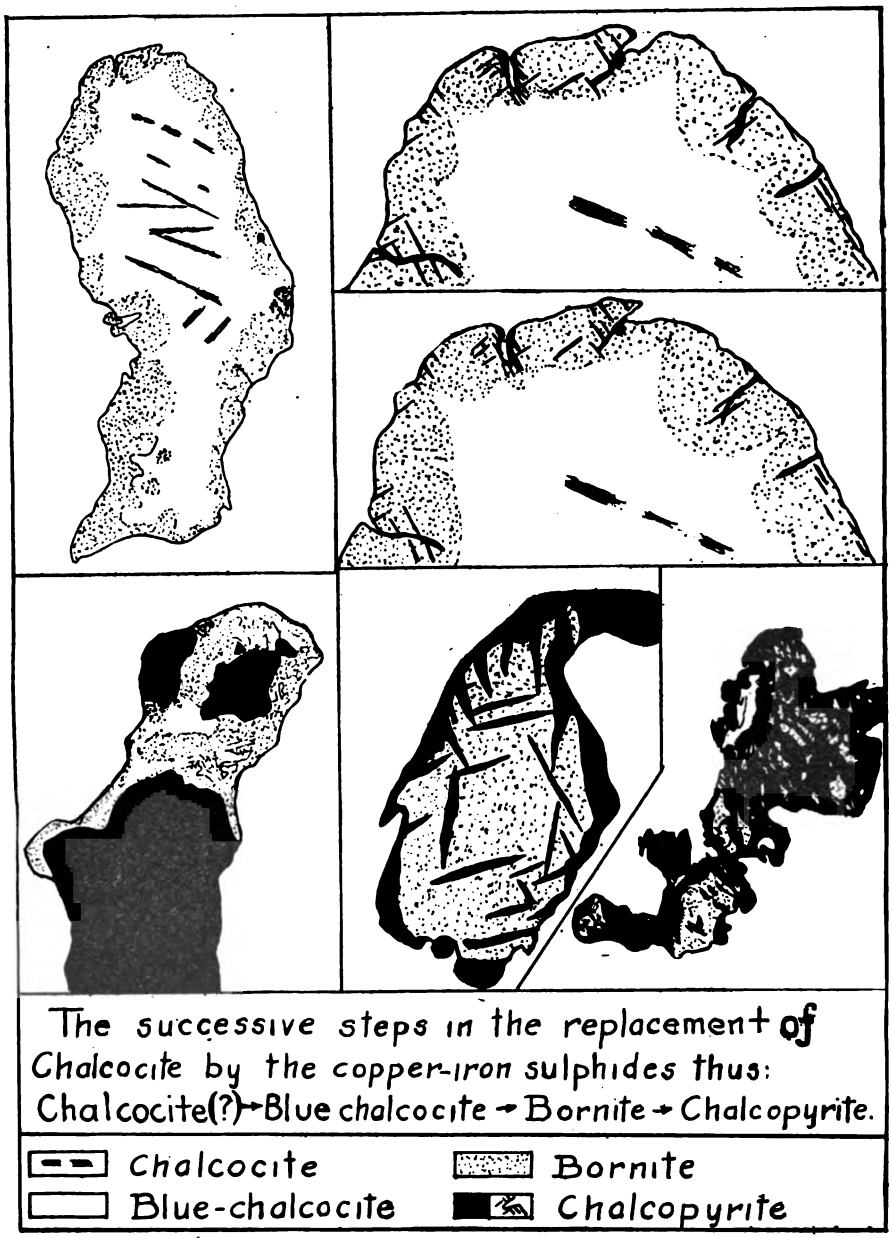

PLATE VIII

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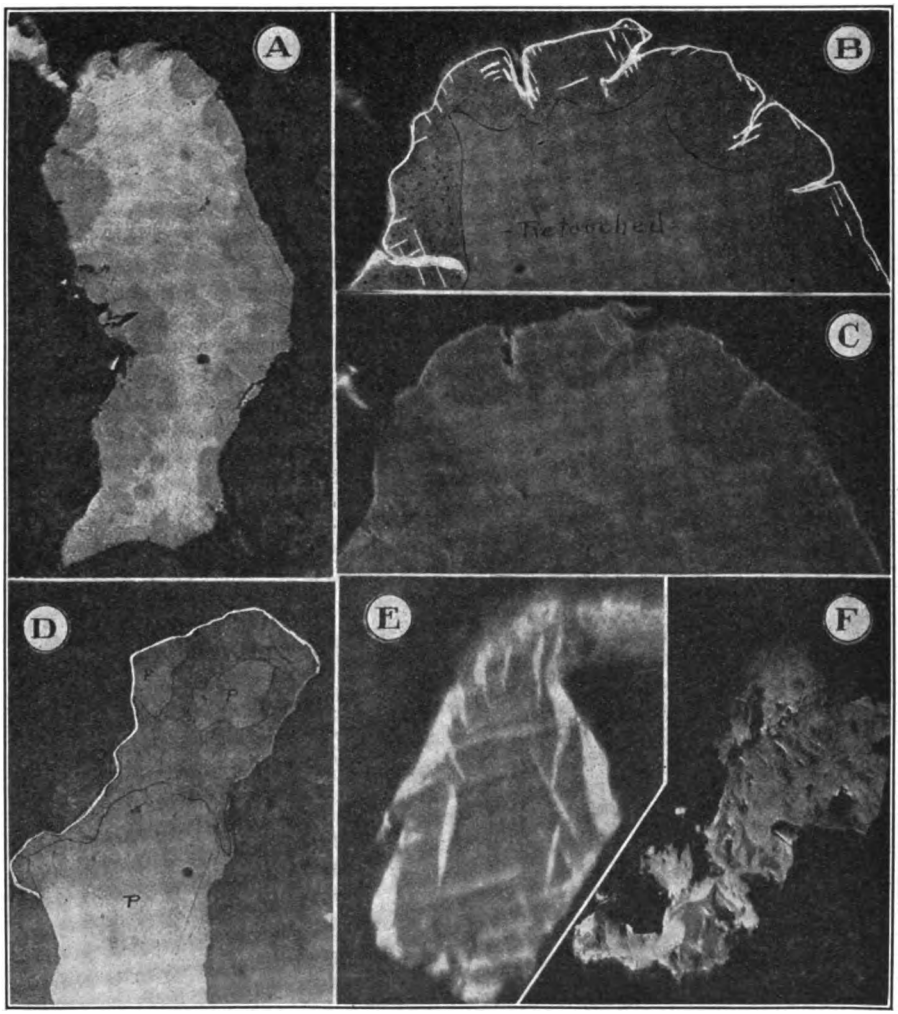

Chalcopyrite Replacing Chalcocite, as Borders and Parallel Gashes, showing Partial to Almost Complete Replacement.

A.—Partial Replacement of Chalcocite by Bornite and Chalcopyrite. × 111 diameters.

B, C.—Enlargements of *A*. Chalcocite Ores. × 341 diameters.

D.—Primary and Secondary Chalcopyrite. × 63 diameters.

E.—Secondary Chalcopyrite Replacing Secondary Bornite. × 431 diameters.

F.—Secondary Chalcopyrite almost completely Replacing Bornite and Chalcocite. × 95 diameters.

PLATE VIII

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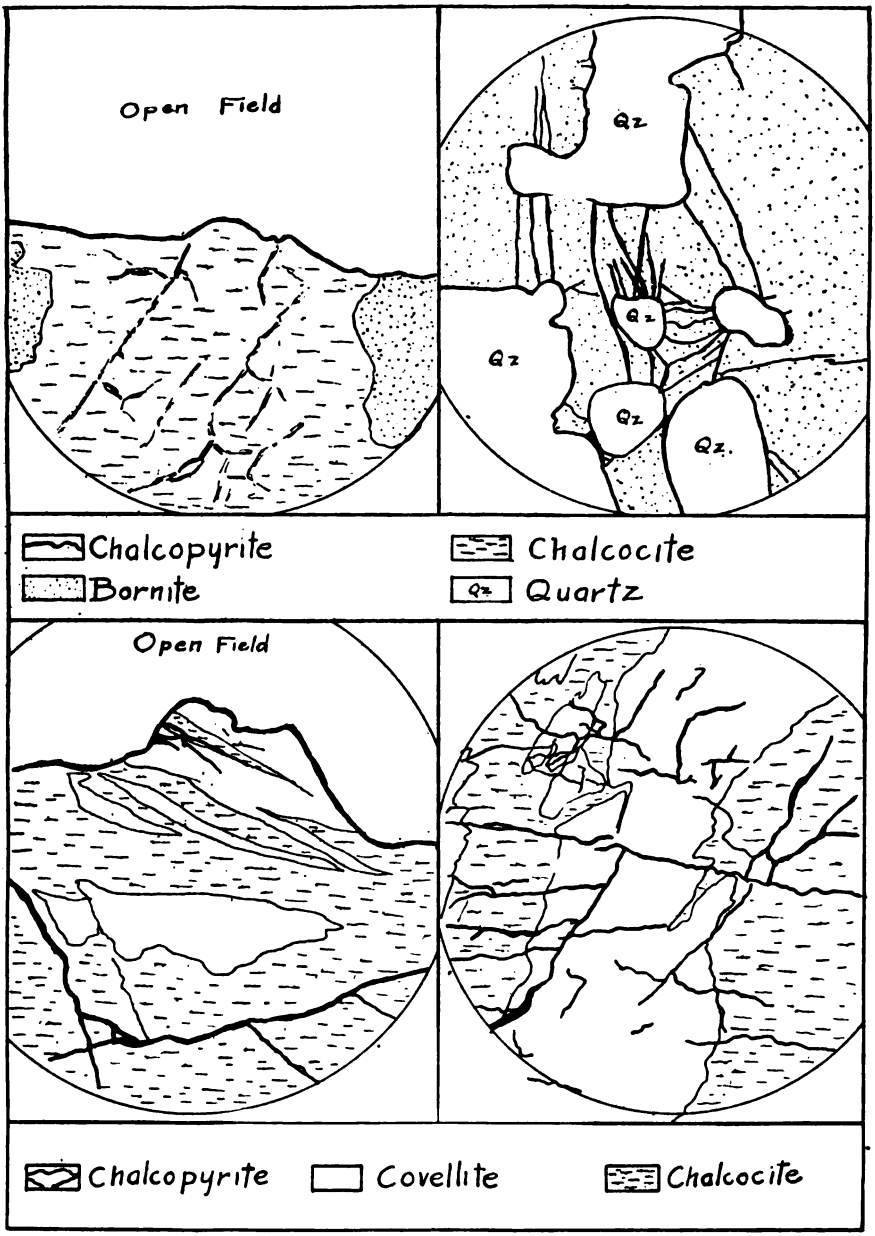

PLATE IX

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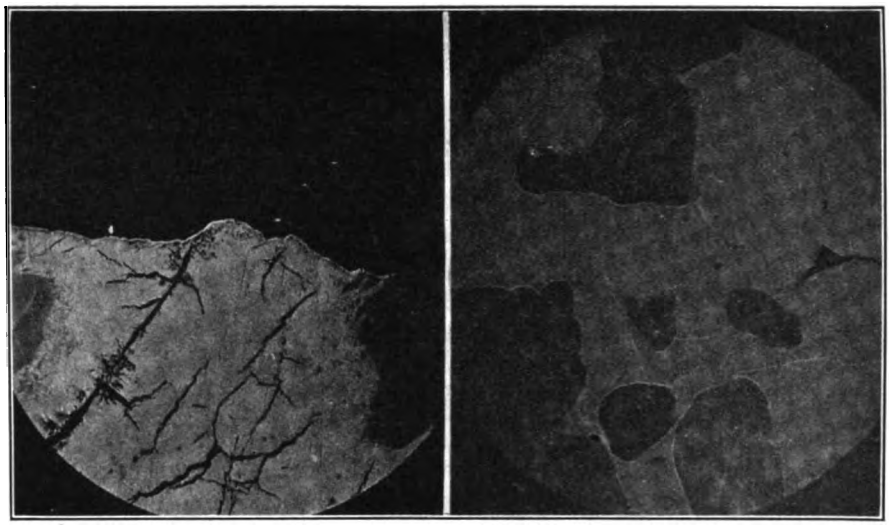
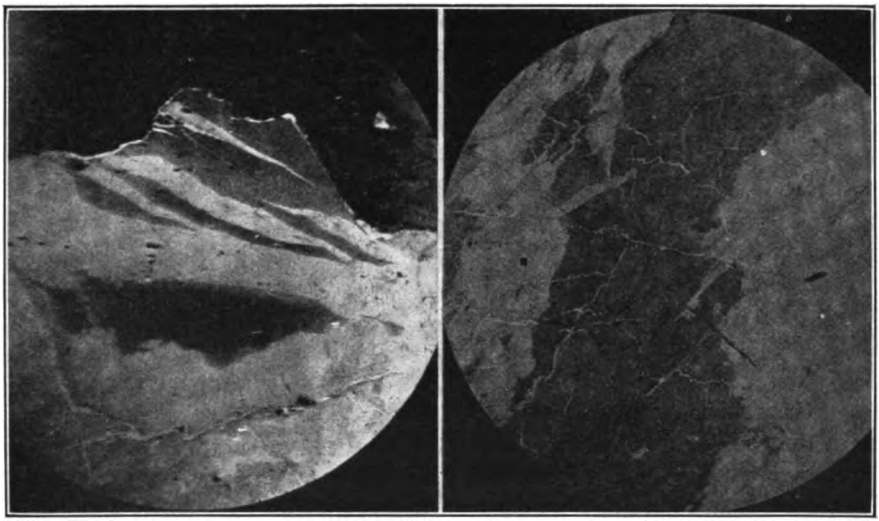


A.—Synthetic Chalcopyrite Replacing Chalcocite. × 57 diameters. Polished section.

B.—Synthetic Chalcopyrite Replacing Bornite in Veinlets and Bordering Quartz Grains. × 51 diameters. Polished section.



C.—Synthetic Chalcopyrite Replacing Chalcocite and Covellite. × 59 diameters. Polished section.

D.—Synthetic Chalcopyrite Replacing Covellite and Chalcocite. × 55 diameters. Polished section.

PLATE IX

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deposits in the limestone lentils, and fissure veins; but the orebody producing more copper than the combined output of all other mines in the district is located in one of the larger masses of quartz-monzonite and is of the disseminated type.

e. The Utah Copper Mine.—In the mass of monzonite porphyry, locally termed the Bingham laccolite, forming the larger part of a mountain, about 1,600 ft. high, between Bingham and Carr Fork Canyon, the development work of the Utah Copper Co. has proved the existence of an enormous orebody. A description of this property and its development is given in the following quotation, from a report issued in August, 1914:[2]

"The orebody of the Utah Copper Co. consists of an altered siliceous porphyry, containing small grains of copper minerals, very uniformly disseminated throughout the mass, both in fracture seams and in the body of the rock, averaging about 1.5 per cent. copper, 0.15 oz. of silver and 0.015 oz. of gold.

"The total area of lode mining claims in Bingham owned by the Utah Copper Co. is 736 acres, within the boundaries of which development has shown that at least 225 acres contain mineralized porphyry of commercial value. The entire porphyry area has not yet been developed; and the maximum thickness of the orebody in the 225 acres has not been fully demonstrated, but the existing development in this area shows now an average thickness of 445 ft., which is equivalent to about 1,500,000 tons of ore per acre, or a total of about 361,220,000 tons. In making the calculations to determine the tonnage of ore and its average assay, there were used 50,761 assays, representing 23,465 ft. of diamond-drill and churn-drill holes, 285,913 ft. of drifts, raises, and winzes, and 7,130 linear feet of steam-shovel cuts, or a total of 316,508 linear feet of development work. The orebody, as at present developed, has a maximum length of a little over one mile and a maximum width of more than one-half mile. Further development of the property will add materially to its ore reserves."

From 1906 to 1913 this property was developed on a large scale by steam-shovel and underground methods. In 1913, the underground work was discontinued, and since that time the entire production has been handled with the steam shovels.

The steam-shovel output has increased from 300 tons of ore daily in 1906 to 22,000 tons daily in 1914. At present 21 steam shovels are engaged in the development work. Some of the shovels are working in ore and others are removing capping. In the future the number of shovels working in the ore will be increased, while those working in capping will be decreased; thus the daily production will gradually increase as the work proceeds.

f. Field and Laboratory Work.—In 1912, the writer was employed by the Utah Copper Co. for six months as underground sampler, and in the summer of 1914 three months were spent on the surface. During this time a suite of about 300 specimens was collected. This collection in-

[2] American Institute of Mining Engineers: One hundred and eighth meeting, Salt Lake City, Utah, Aug. 10 to 14, 1914.

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cludes surface and underground samples of the quartz-monzonite from many parts of the district in and outside the area of mineralized porphyry.

From a selection of the material collected the writer has made and studied—in the Laboratory of Economic Geology at Stanford University—70 thin sections and more than 100 polished sections, with the view of determining the possible stages through which the copper has passed from the time it existed in the earliest magma to its present condition as disseminated copper minerals, enriched by secondary processes.

2. *The Quartz-Monzonite*

The quartz-monzonite in the hand specimen is usually a dark-gray, fine-grained, or light-gray porphyritic rock, containing locally a few exceptionally large phenocrysts of orthoclase—½ to 1 in. long. *A* and *B*, Plate I, ear photographs (natural size) of hand specimens of the mineralized porphyry. The photographs show both the fine and coarse textures of the porphyry, even after intense silicification. The finely crystalline porphyry—*A*—predominates throughout the district; but locally the coarse-textured rock may be present. In general, thin sections of the unaltered quartz-monzonite show uniformity in mineral constituents regardless of the texture.

The chief minerals of the quartz-monzonite, named in the order of their abundance, are orthoclase, and plagioclase, occurring in well-developed phenocrysts and in the granular groundmass; biotite, in phenocrysts and in small flakes; augite, often surrounded by alteration rims of uralitic hornblende (*A*, Plate II), and interstitial quartz.

The accessory minerals are magnetite, in large and small irregular grains, and rutile, occurring as sagenite webs in biotite. Apatite and zircon are present in very small well-formed crystals.

A and *B*, Plate II, are photographs of thin sections of the quartz-monzonite. *A* is from a sample collected on the mountains above the Silver Shield mine; *B* is from an outcrop near the Highland Boy mine. These localities are about 1 mile apart and both are about 1 mile from the disseminated ore deposit. In *A* the plagioclase is more prominent than orthoclase; while in *B* orthoclase is the chief mineral in the groundmass. The illustrations show the principal minerals; also the association of biotite with magnetite.

The interesting features of the rock are the association of biotite with the magnetite, and the occurrence of interstitial quartz.

The magnetite is present in larger amounts than is common in igneous rock of the monzonitic type. It occurs as large and small anhedral grains, often in biotite crystals or surrounded by a narrow margin of small crystals of this mineral. A careful study of these margins shows that they have assumed their present position later than the solidification of the feldspar groundmass. There are two possible interpretations of

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this phenomenon: (1) the biotite was formed by a reaction between the magnetite and certain late magmatic products; (2) the magnetite is of a late magmatic period and is replacing the biotite. The fact that the same phenomenon was observed in slides from the Sudbury district and was evidently caused by a partial replacement of the biotite by the sulphides, inclines the present writer to favor the second interpretation. In either case, this occurrence of biotite and magnetite explains the apparent replacement of biotite by the sulphides in the hydrothermal period, as will be explained later.

The interstitial quartz occurs in spaces about previously formed crystals and was the last silicate mineral to solidify. It is present in all slides, but usually not in large amounts. However, when the amount is large, certain changes take place; in some slides the phenocrysts of feldspars appear to have been attacked and changed into the finer-grained granular feldspars of the groundmass, while the augite shows alteration rims of hornblende. A further concentration of this quartz in the form of small pegmatite veins produces similar changes but to a greater degree.

3. *The Pegmatite Veins*

Specimens of porphyry from the Commercial mine—outside the area of mineralized porphyry—contain small white veinlets, ½ in. in thickness, composed of quartz and feldspar which are visible to the naked eye. The large amount of quartz in the country rock gives it the appearance of a granite rather than of the typical quartz-monzonite.

Examination of the thin section under the microscope shows an increase in the interstitial quartz, accompanied by the formation of a graphic intergrowth of quartz and orthoclase, as shown in *C*, Plate II. This intergrowth occurs on the edge of the veinlet, while the main body of the vein is composed of irregular grains of quartz and orthoclase. The pyroxene has been completely changed to hornblende, which still possesses some of the petrographic properties of the original pyroxene. Biotite has almost completely disappeared and epidote is present instead. Rutile is absent and titanite seems to have taken its place. The magnetite and pyrite are present in separate and connected anhedral grains but not in amounts that would indicate a considerable addition of iron. The magnetite appears to have been attacked and it is possible that hydrogen sulphide was responsible for the change of magnetite to pyrite. The polished section shows the presence of a very small amount of chalcopyrite.

The relation between the pegmatite and the high-temperature quartz vein was not apparent in the field, so only a limited number of specimens were taken; however, the similarity in the mineral constituents suggests a close relation between the two.

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4. *The High-Temperature Veins*

A sample from the same location as the pegmatite veins—according to Lindgren's Mineral Classification—proved upon microscopic examination to be a high-temperature vein.

The vein was composed of massive quartz containing a considerable amount of metallic sulphides. A narrow strip of green hornblende separated the quartz and the country rock.

Examination of the thin section showed the hornblende strip to be composed of hornblende, formed from pyroxene, together with considerable amounts of epidote, titanite, and apatite. The main body of the vein is composed of quartz in which magnetite and pyrite, with a small amount of chalcopyrite, occur in such relations that they do not appear to have replaced any of the silicate minerals or to have replaced one another. It is very probable that they assumed their present form upon the solidification of the quartz.

An interesting feature of the quartz is that many of the crystals are clear in the center but in the outer zone contain many small inclusions which appear to be apatite and zircon. *D*, Plate II, shows the contact between the hornblende strip and the quartz; also the occurrence of the minerals which have been described.

In the Silver Shield mine and also in the Sulphide mine, near the Highland Boy, samples of the porphyry near certain veins showed alterations similar to those which have just been described, except quartz, titanite, and apatite, which were practically absent, while biotite and epidote were both present. In some slides the biotite and pyroxene were both altering to chlorite.

The pyrite and chalcopyrite are not magmatic sulphides, because secondary quartz, epidote, or other minerals,—which indicate the action of mineralizers,— are present in the same slide.

The vein minerals and rock alterations which have been described are common in the high-temperature veins, in the quartz-monzonite, in different parts of the district, but are not found in the disseminated ores; furthermore, the alterations which are found in the disseminated ores are characteristic of hydrothermal conditions; thus there is little doubt that the disseminated ores were deposited by heated aqueous solutions rather than under gaseous or magmatic conditions.

The pegmatite and the high-temperature quartz vein, from the Commercial mine, may be considered as rock extracts, and indicate, to a certain degree, the composition of the mineralizers which were active during the solidification of the magma.

The processes which have been effective in the formation of these

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mineralizers and also in the bringing about of the alterations which have been described are clearly given in the following words by Harker.[3]

"Pneumatolytic Action.—With the completion of crystallization in a rock-magma the contained water and other volatile substances, excepting such portion as has been incorporated in some of the crystallized minerals, must be disengaged. Since the critical temperature for water is about 365 degrees centigrade, and for other substances lower, they must be in the gaseous state, however great the pressure to which they may be subjected.

"The active rôle of the volatile constituents does not terminate with the completion of the crystallization. Having fulfilled the constructive office of mineralizers, they now enter upon a new activity which is partly of a destructive kind. As cooling proceeds, some of the compounds crystallized at higher temperatures, cease to be stable in the presence of the concentrated gaseous residuum, and are decomposed with the production of new minerals. The volatile substances themselves may or may not enter into the composition of these new minerals, but in general they do so to a greater extent at this stage than during the magmatic crystallization.

"Pneumatolytic action in plutonic rocks is not confined to their pegmatoid modifications. In many cases it clearly follows joint-fissures in the body of the rock, sufficient proof that it belongs to a stage distinctly posterior to the completion of consolidation."

At lower temperatures these gases become liquids and the changes due to hydrothermal activity follow.

5. *The Early Hydrothermal Period*

The early part of the hydrothermal period marks the most important period of mineralization in the disseminated ores, because the solutions in this period caused the comparatively uniform dissemination of chalcopyrite and pyrite in an area more than 1 mile long and in some places more than ½ mile wide.

Although faulting and intense fracturing in the rock are widespread throughout the district it does not appear that in the disseminated area the early mineral-bearing solutions were confined to fractures or fissure veins, but, on the contrary, these solutions have caused exceptionally uniform distribution of fine-grained secondary quartz and sulphide grains throughout the entire mass of the rock in large areas.

The following quotation from Mr. Boutwell's report describes clearly the mineralized quartz-monzonite of the disseminated ores and also suggests the possible relation between the magnetite, chalcopyrite, and pyrite.

"A specimen of highly altered monzonite has lost the dark color and compact body and shows instead a dull, light gray color, a slightly porous structure, and abundant quartz in veinlets and blotches upon the walls of parting planes. Conspicuous areas of granular quartz are numerous, the orthoclase is highly sericitized and femic minerals

[3] Alfred Harker: *The Natural History of Igneous Rocks*, p. 266 (1909).

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are represented by numerous irregular patches of small individuals or flakes of dense brown biotite. The quartz and sericite are clearly secondary and though no direct proof of the age of the biotite has been found, it resembles secondary biotite and may be secondary also. Magnetite, excepting occasional grains, has disappeared, and large amounts of chalcopyrite and pyrite are present in the form of rounded grains, chains, and veinlets embedded in secondary quartz, flaky biotite, and sericitized feldspar.

"The observed stage of metasomatic alteration of magnetite, culminating in the occurrence of minute, ill-defined cores of magnetite without secondary sulphides and finally in the total disappearance of magnetite, indicates one source of iron of the chalcopyrite and pyrite. Additional iron was doubtless derived from original augite and biotite. Any additional sulphur which may have been required was probably supplied from without, perhaps in the form of sulphurous gases."

The alterations just described and the conclusions drawn in this quotation are in harmony with the present paper.

A detailed study indicates that the first alterations were as follows: The feldspar groundmass in large areas of the quartz-monzonite has suffered complete metasomatic replacement by a very fine-grained secondary quartz, the feldspar phenocrysts have been partly, and in some cases completely, replaced by this secondary quartz; but in spite of this intense silicification some of the remaining feldspar fragments are perfectly fresh and free from sericite, while others have varying amounts, which may have been formed later than this period. The sericite which is present in the slides showing the least amount of this mineral is confined to the feldspars and does not replace the secondary quartz. There are two possible explanations for the small amount of sericite: (1) secondary biotite may have been formed instead of sericite; (2) the amount of carbon dioxide in the first period of mineralization was either small or inactive.

1. In the mineralized monzonite least affected by sericitization there are usually large amounts of small green and brown crystalline aggregates of secondary biotite. This biotite may have formed instead of sericite because of the excess of iron present, part of which was probably in the form of more or less complex molecules which formed in the break-down from the original augite, which in this altered rock has completely disappeared, leaving no direct evidence as to its alteration products. In this case the secondary biotite may have the formula of sericite in which part of the aluminum has been replaced by iron, giving the formula $Al_2Fe(SiO_4)_3KH_2$, described by Clarke.[4] However, in many slides the secondary biotite is clearly derived from the primary biotite—*B*, Plate III, which is broken up into smaller crystals, many of which are still oriented in parallel position in the space occupied by the primary crystal, and probably have the same relation to the primary biotite as sericite has to muscovite.

[4] F. W. Clarke: The Constitution of the Natural Silicates, *Bulletin No.* 588, *U. S. Geological Survey*, p. 51 (1914).

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2. The formation of sericite often assumes the presence of carbon dioxide; its formation from orthoclase may be represented by the following equation:[5]

$$\underset{\text{Orthoclase}}{3KAlSi_3O_8} + \underset{\substack{\text{Carbon}\\\text{dioxide}}}{CO_2} + \underset{\text{Water}}{H_2O} = \underset{\text{Sericite}}{KH_2Al_3(SiO_4)_3} + \underset{\substack{\text{Potassium}\\\text{carbonate}}}{K_2CO_3} + \underset{\text{Quarts}}{6SiO_2}$$

In the early hydrothermal period the amount of carbon dioxide was either small or in the presence of the other components of the solution it was not effective in producing sericitization. Lindgren in speaking of volcanic gases says:[6]

"Among the volcanic gases nitrogen, carbon dioxide, and hydrogen sulphide are the most important and their emission, particularly that of carbon dioxide, may continue long after the igneous activity has subsided. . . . Evidence of this is furnished by the exhalations of carbon dioxide and nitrogen in the mines at Cripple Creek, of nitrogen at Creede, and of carbon dioxide in the Tertiary gold deposits of New Zealand."

The alterations in the disseminated ores give strong evidence that sericitization for the most part has been effective after the formation of the fine-grained secondary quartz and also after the deposition of the disseminated primary sulphides, for in many of the slides the feldspars are completely replaced, and small lath-shaped crystals of sericite penetrate into the chalcopyrite (*A* and *B*, Plate V), but seldom into the pyrite grains; thus the sericite which is present in the slides showing the least amount of this mineral is probably of a later period than the first alteration. The author favors the second explanation.

The replacement of the sulphides by sericite was first described by Rogers in a magmatic ore deposit in Plumas County, Cal.[7] The occurrence in Bingham presents a parallel case.

A, Plate III, is a microphotograph showing the irregular cross-sections of comparatively fresh feldspar phenocrysts imbedded in a groundmass of fine-grained secondary quartz. The accompanying outline plate—drawn from the slide—shows the distribution of the secondary biotite and also the other minerals which are present in the rock containing the least sericite.

The sagenite webs of rutile may still be seen in the fragments of primary biotite, but as the alteration proceeds these crystals separate out in small crystalline aggregates and in larger crystals which are evidently formed by recrystallization.

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[5] F. W. Clarke: The Data of Geochemistry, *Bulletin No.* 491, *U. S. Geological Survey*, p. 512 (1911).

[6] Waldemar Lindgren: *Mineral Deposits*, p. 445 (1913).

[7] H. W. Turner and A. F. Rogers: A Geologic and Microscopic Study of a Magmatic Copper Sulphide Deposit in Plumas County, California, and Its Modification by Ascending Secondary Enrichment. *Economic Geology*, vol. ix, No. 4, p. 386 (June, 1914).

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The pyroxene and hornblende have completely disappeared. Occasionally in the porphyry outside of the mineralized areas magnetite appears to form when the pyroxene is undergoing alteration; it is therefore probable that some of the iron present in the sulphides was derived from this source.

Zircon and apatite are more noticeable than in the unaltered monzonite, but are probably unaltered minerals which were present in the original rock.

Magnetite has almost completely disappeared and in its place there are chalcopyrite and pyrite, the distribution of which bears a striking similarity to that of the magnetite in the unaltered quartz-monzonite.

Occasionally pyrite or chalcopyrite occurs in altered crystals of the primary biotite in such a way that the replacement of biotite by the sulphides is suggested. *B*, Plate III, is a good example of this apparent replacement, but it will be noted here that a small crystal of secondary biotite, showing no signs of replacement, is present as an inclusion in the sulphide. These inclusions are not uncommon in grains of chalcopyrite and occasionally in pyrite. In some slides the secondary biotite cuts the sulphide grains and also the secondary quartz in such a way as to indicate that it is later than either. Considering the fact that secondary biotite is present in such large amounts forming simultaneously with and probably following the deposition of the sulphides, it is not reasonable to suppose that the biotite has been replaced by the sulphides.

The only conceivable way in which the sulphides could have been so evenly distributed, in a deposit of hydrothermal origin, would have been through the medium of a precipitating agent which was itself disseminated throughout the original quartz-monzonite. This precipitating agent was evidently confined to the quartz-monzonite because in the quartzite, which has been silicified by the action of the same solutions as have produced silicification in the quartz-monzonite, there is at most only a narrow margin which is mineralized.

In the Physical Chemistry Laboratory of Stanford University recent work has shown that alkaline solutions charged with hydrogen sulphide, alone or in the presence of carbon dioxide, under pressure are capable of carrying considerable amounts of metallic sulphides in colloidal suspension; furthermore, upon the escape of hydrogen sulphide the metallic sulphides are precipitated. A discussion of the experiments together with the geologic evidence supporting these results appears in a recent paper by Tolman and Clark.[8]

In *C*, Plate III, the sulphides appear along the cleavage planes of the primary biotite. The polished section *D*, Plate III, highly magnified,

[8] C. F. Tolman, Jr., and J. D. Clark: The Oxidation, Solution, and Precipitation of Copper in Electrolytic Solutions, etc. *Economic Geology*, vol. ix, No. 6, pp. 559 to 592 (Sept., 1914).

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shows the sulphides in the biotite to be pyrite in small grains, and covellite and chalcocite replacing the primary chalcopyrite which was deposited in small gashes along the cleavage planes of the primary biotite. Occurrences of this kind are rare but they certainly suggest a precipitating agent within the biotite.

Comparison of *C*, Plate III, with *B*, Plate II, shows the similarity in the occurrence of pyrite in biotite—in the mineralized area—to that of magnetite in biotite in the unaltered monzonite. Occurrences of this kind together with the similarity in the distribution of magnetite in the unaltered rock, and the sulphide in the mineralized monzonite, furnish good evidence that the magnetite is responsible for the precipitation of the sulphides, and also explain the apparent replacement of biotite by pyrite and chalcopyrite.

The following equation, though experimental work tends to prove it,[9] is given merely to suggest the possible reactions which have taken place in the heated alkaline solutions carrying the copper and probably some iron as colloidal sulphides:

$$\underset{\text{Hydrogen sulphide}}{4H_2S} + \underset{\text{Cupric sulphide}}{2CuS} + \underset{\text{Magnetite}}{Fe_3O_4} = \underset{\text{Chalcopyrite}}{2CuFeS_2} + \underset{\text{Pyrite}}{FeS_2} + \underset{\text{Water}}{4H_2O}$$

CuS is used here merely to balance the equation, but another copper or copper-iron sulphide could be substituted.

In the foregoing equation the removal of hydrogen sulphide from the solution and the formation of the less soluble sulphides chalcopyrite and pyrite, have both been factors in the precipitation. Magnetite has furnished iron for both chalcopyrite and pyrite, but there may have been an addition of iron at this time.

Another possibility is that the change of magnetite to pyrite by the action of hydrogen sulphide—proved experimentally by Doelter[10] and others—has removed hydrogen sulphide from the solution, thus causing the precipitation of chalcopyrite directly. In either case there has evidently been a molecular rearrangement of the iron, because biotite inclusions are present in both pyrite and chalcopyrite.

In many of the slides and polished sections chalcopyrite and pyrite occur in separate grains in such a way as to indicate that the pyrite has not been replaced by chalcopyrite—for instance, in a certain area, say there are 12 sulphide grains; five may be composed wholly of pyrite, two of pyrite with a partial border of chalcopyrite, and five of solid chal-

[9] Prof. S. W. Young. Personal communication: When chalcocite or covellite and magnetite are placed in the same test tube in slightly acid, alkaline, or neutral solutions in the presence of hydrogen sulphide under pressure the chalcocite or covellite is rapidly coated with chalcopyrite and the magnetite is probably changed to pyrite.

[10] F. W. Clarke: The Data of Geochemistry, *Bulletin No.* 491, *U. S. Geological Survey*, p. 317 (1911).

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copyrite, without any cores or residual fragments of pyrite in the chalcopyrite. These grains have no special arrangement with respect to veinlets or parting planes in the rock. Occurrences of this kind are considered evidence that in certain parts of the deposit chalcopyrite does not replace pyrite. Ores from the "Pit" do show a replacement of pyrite by chalcopyrite; this will be discussed later. After considering other evidence it is not at all impossible that chalcopyrite and pyrite were formed in many parts of the deposit at about the same time. Where chalcopyrite and probably a small amount of bornite appear with but little pyrite it is simply due to solutions richer in copper than those which have formed pyrite with less chalcopyrite and the direct change of magnetite to pyrite has taken place.

It is true that the rocks in all parts of the district are intensely shattered, but the uniformity in the distribution of the fine-grained secondary quartz throughout the mass of mineralized monzonite would suggest that if these fractures existed they were extremely small under the great pressure at depth. Outline Plate I, made from the photographs and polished sections, natural size, shows the distribution of the sulphide grains with reference to the small fractures.

The precipitation of sulphides in the way which has just been described favors a great vertical range for the deposition of the chalcopyrite and pyrite. The present development work of the Utah Copper Co.—surface and deep drilling—has proved this range to be a minimum of 2,500 ft.

6. *The Late Hydrothermal Period*

The late hydrothermal period is probably a continuation of the early period, occurring after a more intense fracturing and faulting throughout the district. Although this period was probably an important one in the mineralization of the lower-temperature replacement deposits and fissure veins bordering the monzonite, it has been of less importance in the disseminated deposit, because in the disseminated ores the small veinlets, though quite numerous, seldom exceed 2 or 3 in. in thickness and are never included in the regular mine samples. With these sulphides, coveilite or chalcocite is generally present in such amounts that it is not possible to determine with certainty the order of formation of the different minerals.

Molybdenite is common as very thin coatings on slickensides and parting planes in the porphyry. This mineral is usually formed at high temperature and may have been formed in the early hydrothermal period.

Enargite, sphalerite, and galena are rare. To the writer's knowledge there are only four places in the entire area of mineralized porphyry where these minerals are found in veinlets.

Typical examples of some of the small veinlets together with enlargements from their polished sections are shown in *A*, *B*, *C*, and *D*, Plate IV.

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The alterations accompanying and following this later period of mineralization have been silicification and sericitization along the parting planes in the rock. The intense silicification usually extends not more than an inch on either side of the veinlets in the black porphyry, but in the workings of the underground mine, located below the top of the mountain, a strip of monzonite about 200 ft. thick, bordering the quartzite, has been so intensely silicified and sericitized as to produce a light-colored rock which has been thought to be a separate intrusive mass. This light-colored rock is locally termed the "Payroll porphyry," after the Payroll group of claims, and hereafter will be called by that name. The Payroll porphyry outcrop forms the top of the mountain, where its resistance to weathering has governed the topography to a considerable degree.

The light color and the small brown specks visible to the naked eye in the hand specimen, serve to distinguish the Payroll from the "Black" porphyry—altered quartz-monzonite containing a large amount of primary and secondary biotite.

Examination of thin sections of Payroll porphyry proves beyond doubt that it is the same porphyry as the darker quartz-monzonite, which it grades into without a well-defined contact. The light color is due to the large amount of quartz and sericite, while the brown spots are shown to be small aggregates of rutile crystals occurring in bleached biotite or in the spaces once occupied by the primary biotite crystals.

While the later period may not have been important as one of mineralization, the sericitization in this period has been of great importance in increasing the porosity of the rock, in the Payroll porphyry throughout the entire mass, and in the Black porphyry along numerous small fractures in the rock, thus promoting uniform enrichment by surface waters.

II. Secondary Enrichment

1. *Author's Note*

In the following pages the term secondary will be used in the sense of changes, alterations, and enrichment caused by descending surface waters or solutions.

The reader will keep in mind that the photographs of the polished sections in this paper, as in most of the recent papers, show areas which in the natural size are extremely small (compare sulphide grains, Plate I, with enlarged grains, Plate V, etc.) and thus the replacements shown are on a very small scale.

The conventional design on the outline plates indicating a certain mineral has been retained for that mineral, so far as possible, in all of the plates. The drawings have been made from the photographs and also by the use of the microscope and are therefore accurate. No attempt has been made

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to show the different silicate minerals bordering the sulphide grains in the polished section, because determination of these minerals by reflected light is at best only a guess. In this paper determinations of silicate minerals have been made in thin sections. In the thin section the sulphide grains are usually surrounded by fine-grained secondary quartz (shown in *A*, Plate III), small flakes of green and brown biotite, and in most of the specimens a large amount of sericite is present.

2. *Factors Governing Enrichment*

The principal factors governing the enrichment of the disseminated copper ores in Bingham Canyon are as follows:

(*a*) The intense fracturing of the rock has formed numerous channels, affording uniform circulation of surface waters. *A*, Plate I, shows this fracturing, also faulting on a small scale. Often it is difficult to obtain hand specimens of the ordinary size without many small fractures running through them.

(*b*) Sericitization has increased the porosity, and thus the permeability to surface waters, along the fractures and in large areas throughout the entire mass of the rock.

(*c*) Erosion has taken place more rapidly than oxidation because, as the canyons have been cut on each side of the mountain, rather steep soil-covered slopes have been formed; the impermeability of these slopes has caused a large part of the surface water to run down the mountain side, or soak into the soil for only a short distance; thus only the water from heavy precipitations—rain water or that from thawing snow—has passed down through the rocks. Oxidation has been slow, and only a comparatively thin blanket of oxidized leached capping covers the orebody. Variation in the thickness of the capping is governed by the topography of the mountain and the permeability of the rock.

(*d*) In recent geologic periods, at least, the elevation of the disseminated ores has been high—at present about 7,300 ft.; therefore cold winters and warm summers have doubtless governed to a certain degree the rate of oxidation.

3. *The Enrichment*

The occurrence of the two principal secondary copper minerals divides the enrichment of the disseminated ores into two different types: (1) where covellite is more abundant than chalcocite; (2) where chalcocite is more abundant than covellite. Although the ores of these two types blend into each other they are still distinct enough to be taken up under separate headings.

The greatest depth to which the enrichment extends is about 800 ft. below the surface, but the average thickness of the entire known orebody is 445 ft., and it is covered by an average thickness of capping of 114 ft.

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4. *The Covellite Enrichment*

The ore in which covellite is the most important secondary mineral is located below the top of the mountain and upper levels. A rough boundary might be taken to include the surface and underground ore above Sub-level 5 of the underground mine, as shown on the longitudinal section, Fig. 1.

In the summer of 1914 the writer spent three months working on the R level. Observations made here will hold good for the levels from the top of the mountain to a number of levels below R.

The steam shovels and drills along some of these levels are working part of the time in ore and part of the time in capping; thus a good opportunity is afforded for studying both.

(*a*) *The Capping above the Covellite Ores.*—The capping is usually composed of several feet of soil and 50 to 100 ft. of leached porphyry. The leached rock is colored with the characteristic browns of the iron ores. Near the soil the brown color is lighter, but passing downward it shades into a darker brown until, in a transition zone of a few feet, the rock assumes the dark-gray color of the ore. When the bank is freshly broken by a blast these colors are most noticeable; after a few days the moisture dries out, and the brown fades, but the gray appears wet, due to the presence of sulphuric acid, together with copper and iron salts. A short time after this surface is exposed a greenish stain appears on the gray rock, extending from 5 to 20 ft. below the brown, and in the course of a few days the evaporation of moisture causes a thin coating of copper sulphate crystals to form.

On the R level a churn drill was used to drill 6-in. holes for blasting purposes. The holes were drilled along the edge of the bank, 40 ft. apart, and to an average depth of 45 ft. While drilling in the brown capping the bit was scoured and clean, but when working near the contact of the ore green spots appeared on it, and in the ore just below the surface or brown capping it was often covered with a visible coating of copper.

The latter is remarkable because while the drill was in operation the scouring action of the grit in the slime would prevent the formation of this coating; therefore it must have been formed between the stopping and the hoisting of the bit, which at most was not more than 4 or 5 min., thus indicating a fairly concentrated solution of copper sulphate.[11]

In general, the observations point to the fact that during the warm dry months of the summer, when the rock is moistened only by an occasional shower, oxidation is rapid and the iron and copper sulphates,

[11] It must be understood that this rock was only slightly moist, but while drilling enough water was poured into the hole to make a fluid slime and the copper sulphate was dissolved in this water.

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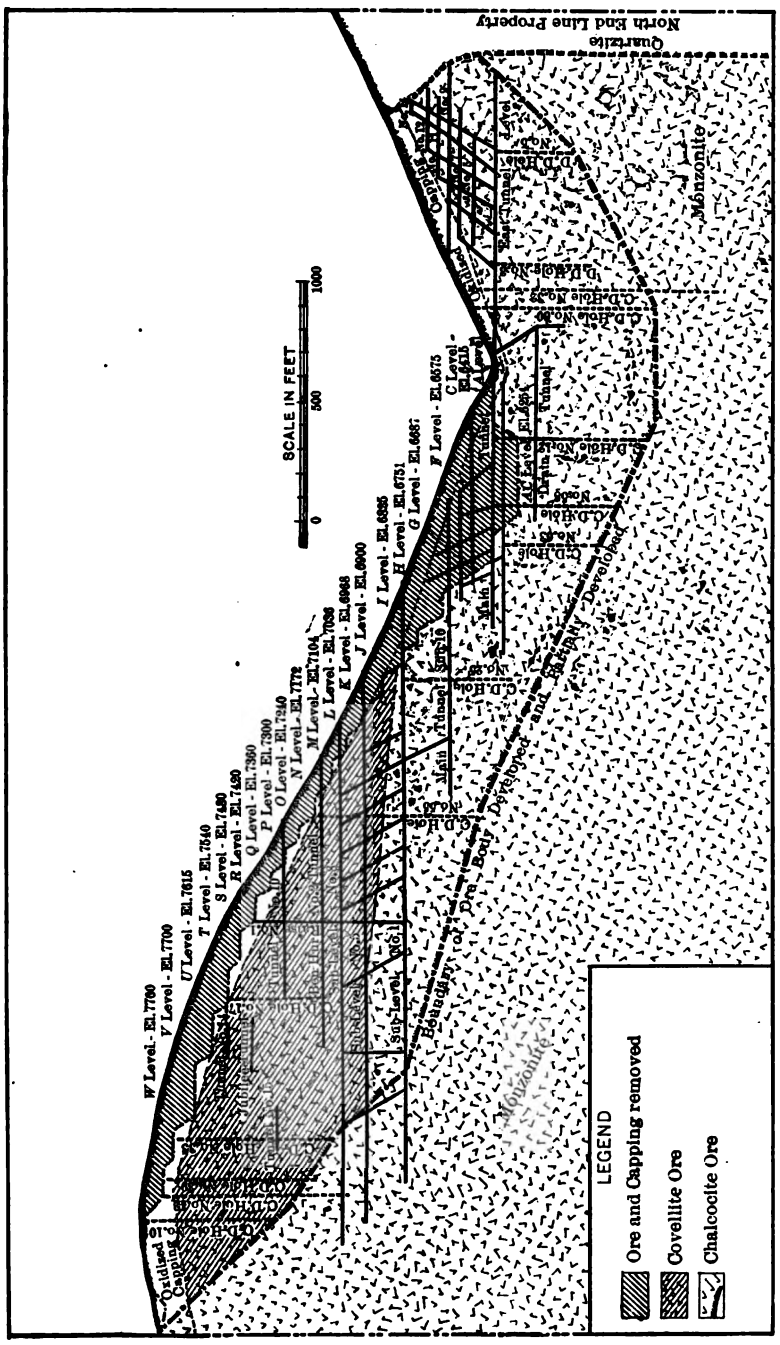

FIG. 1.—LONGITUDINAL SECTION THROUGH CENTRAL PORTION OF OREBODY OF THE UTAH COPPER CO., BINGHAM CANYON, UTAH.

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together with sulphuric acid, accumulate in considerable amounts just below the capping.

The water from the first heavy precipitation, percolating through the rocks, dissolves the accumulated salts, producing a more concentrated solution than those which follow for some time. In regions similar to Bingham, where the snow lies on the mountains for a number of months during the winter, oxidation may be slow during the cold weather and thus the solutions which pass down through the rocks will be almost barren of copper and iron salts—possibly until about the same time in the next year. In this case it would be remarkable if the periodic solutions were always of uniform acidity and always carried the same amounts of copper and iron sulphates in solution.

If the secondary minerals showed no signs of periodic changes little would be gained by discussing conditions which would produce changes in the descending solutions, but the disseminated ores do show these changes to a marked degree, as will be shown under the heading "Secondary Chalcopyrite and Bornite."

Another interesting phenomenon which was noted in connection with the churn-drill holes was as follows: In the brown capping a candle burned brightly while being lowered to the bottom of the hole, but in the ore the flame was extinguished a few feet below the surface, or the brown capping. Running the bailer up and down in the hole did not mix the gas and air in such amounts that the condition was improved. No odor of sulphur gases could be detected. It is therefore probable that the oxygen of the air was being rapidly used up, probably by the ferrous sulphate thus:

$$\underset{\text{Ferrous sulphate}}{2FeSO_4} + \underset{\text{Sulphuric acid}}{H_2SO_4} + \underset{\substack{\text{Air}\\ 21+\%\ O_2\\ 78+N_2}}{O_2} = \underset{\text{Ferric sulphate}}{Fe_2(SO_4)_3} + \underset{\text{Water}}{H_2O} + N_2.$$

From this it would appear that the ferric sulphate had been reduced to ferrous sulphate a short distance below the capping. Whether this be so or not this phenomenon would indicate reducing rather than oxidizing conditions. It is usually supposed that the ferric sulphate may extend some distance below the oxidized zone, but if the sulphuric acid does become concentrated just below the brown capping, its tendency to produce hydrogen sulphide by reactions with certain sulphides would be increased. The hydrogen sulphide thus generated would rapidly reduce the ferric to ferrous sulphate thus:[12]

$$\underset{\text{Ferric sulphate}}{Fe_2(SO_4)_3} + \underset{\text{Hydrogen sulphide}}{H_2S} = \underset{\text{Ferrous sulphate}}{2FeSO_4} + \underset{\text{Sulphur}}{S} + \underset{\text{Sulphuric acid}}{H_2SO_4}$$

An excess of this hydrogen sulphide under low pressure might precipi-

[12] W. H. Emmons: The Enrichment of Sulphide Ores, *Bulletin No. 529, U. S. Geological Survey*, p. 91 (1913).

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tate both iron and copper sulphides. The remaining hydrogen sulphide, together with that which may be generated at a greater depth, passes downward with the sulphate solution, carrying with it small amounts of colloidal copper and iron sulphides. Sphalerite is one of the sulphides which produce hydrogen sulphide most readily by the action of sulphuric acid, but it has been found experimentally[13] that under a pressure of two atmospheres the reaction ceases or becomes extremely slow; thus this reaction would take place near the surface, but at comparatively shallow depth the sphalerite may remain unchanged in the presence of acid solutions. Other sulphides will build up greater pressures, but in general the generation of hydrogen sulphide by descending acidic surface solutions is probably a reaction which takes place a short distance below the surface in fairly concentrated acid solutions.

The minerals present in the capping are shown in thin section to be a large amount of secondary quartz which the surface waters have not affected; sericite is partly changed to kaolinite, and often the sericite and kaolinite fill the space once occupied by the original feldspar crystals. In some slides biotite has completely disappeared, leaving only small yellow crystals of rutile in its crystal outline; in other slides the biotite is bleached until it has the appearance of muscovite. The sericitization and bleaching of the biotite is largely a hydrothermal alteration, but the kaolinization is undoubtedly an alteration produced by acidic surface waters. The carbonates, azurite and malachite, are noticeably absent along the upper levels. The sulphides when present in the capping are often surrounded by the brown and yellow oxides of iron. One occurrence of gypsum was noted on the R level, but this was confined to small veinlets in a clay-like gangue about 30 ft. thick.

Aside from the kaolinite and small amount of gypsum there are no alteration products which would indicate a considerable neutralization of the acid solutions by the silicate minerals. It is therefore reasonable to suppose that the copper minerals in and just below the capping will show the effects of acid solutions to a marked degree while the sulphides at depth will show these effects to less degree.

Polished sections made from ores, taken from the upper levels just below the capping, show the disseminated sulphide grains to be coated with covellite. The coating of covellite is usually much deeper on the chalcopyrite grains than on the pyrite; and when it is found as a coating, partly or completely encircling the pyrite grains, it is usually possible to see fragments of chalcopyrite in the covellite, indicating that the covellite was formed by the replacement of chalcopyrite rather than pyrite.

A, *B*, *C*, and *D*, Plate IV, are photographs of polished sections showing small veinlets, full width, together with the enlargements of small areas

[13] Laboratory of Physical Chemistry, Stanford University.

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of each vein. These veinlets were all collected along the R level just below the brown capping. In these sections, as in others from the same vicinity, covellite is shown to be the predominating secondary copper sulphide.

The formation of covellite from chalcopyrite—*A*, Plate IV—may have been quite simple, as shown in the following equation:

$$\underset{\text{Copper sulphate}}{CuSO_4} + \underset{\text{Chalco-pyrite}}{CuFeS_2} = \underset{\text{Covellite}}{2CuS} + \underset{\text{Ferrous sulphate}}{FeSO_4}$$

But in the replacement of galena—*D*, Plate IV—the reaction was probably more complex and could not be indicated by a simple equation because free sulphur was among the other products. The replacement of enargite by covellite would also represent a more or less complex replacement.

In the hand specimen a small veinlet may appear to be composed of pure chalcocite, but a polished section often shows this veinlet to be composed almost wholly of pyrite with microscopic veinlets of chalcocite passing through the pyrite; an example of this is shown in *B'* and *B*, Plate IV. *B'* shows the vein composed of pyrite while *B* shows the microscopic veinlets of chalcocite.

b. Conclusions Drawn from a Study of the Capping and the Covellite Ores.—There is a transition zone of only a few feet, between the oxidized brown capping, carrying practically no sulphides, and the gray rock containing large amounts of sulphides.

In the brown capping oxidizing conditions exist, while in the gray rock just below it reducing conditions predominate.

The copper and iron sulphides, together with sulphuric acid, accumulate just below the capping during the warm summer months, and are dissolved by the water from the first heavy precipitation after the dry season, producing a solution with a higher concentration of these salts than the solutions which follow for some time.

The solutions producing enrichment are periodic. The relative concentrations of dissolved salts and acid vary in different periods.

The solutions passing downward through the orebody are probably slightly acid to a considerable depth unless neutralized by the reactions with sulphide minerals.

Covellite is the predominating secondary copper mineral, formed under slightly acid conditions, in the porphyry below the upper levels.

c. Covellite Enrichment at Depth.—From the underground mine located directly below the top of the mountain and the upper levels, which have just been described, the writer has collected about 125 samples. These samples were taken on the three main levels, Sub 1, Sub 5, and Sub 8, at varying intervals along the crosscuts, perpendicular to the plane of the longitudinal section (Fig. 1). These crosscuts extend from the quartzite

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on one side, directly across the intrusive nearly to the surface of the mountain on the other side, a total distance of about 2,100 ft. on Sub 8, and a somewhat greater distance on the levels below.

The polished sections of these ores invariably show that the enrichment has been similar to that of the disseminated ores of Ely, Nev., described by Mr. Spencer[14] in the following words:

"Both chalcopyrite and pyrite have been replaced by chalcocite but the coatings of this secondary mineral are commonly much deeper on the chalcopyrite than on pyrite, much of which has not been coated at all."

Covellite, however, is more abundant than chalcocite in the ores of the underground mine and is formed in these ores chiefly from chalcopyrite by slightly acid or neutral solutions carrying a considerable amount of copper and ferrous sulphates. Covellite is often replaced by chalcocite, and in some sections it appears that the reverse order may have taken place.

Typical examples of the ores from the underground mine are shown in *A*, *B*, *C*, and *D*, Plate V. These illustrations also show the replacements which have just been described. In *A* and *B* it will be noted that there are small lath-shaped crystals of sericite penetrating the sulphide grains. These sericite crystals are invariably surrounded by covellite, chalcocite, or the secondary mineral, showing that they are not inclusions but have penetrated the grains from the outside and are therefore later than the primary and earlier than the secondary sulphides. The sericite crystals, although microscopic in size, have formed sub-capillary channels into the sulphide grains, thus permitting a more complete replacement by the secondary minerals. These examples also emphasize the importance of sericite as a mineral which increases the porosity of the rock.

D shows bornite and chalcocite forming the center of a grain, while covellite appears to have replaced part of the chalcocite; then a change in the solutions has caused a part of the covellite to be replaced by a border of chalcocite.

Chalcocite is formed more commonly from bornite—*C*, Plate VI—than from chalcopyrite, but under certain conditions the blue chalcocite may develop directly from the chalcopyrite and in a few small fissure veins white chalcocite may replace sphalerite—*D*, Plate VI. In general, there are probably two factors which govern the formation of chalcocite in the disseminated ores: (1) decreasing acidity in the solutions, probably to the extent of neutral or alkaline conditions; (2) relative concentration of copper sulphate to ferrous sulphate; the latter will be discussed under the heading "Secondary Chalcopyrite and Bornite."

[14] Arthur C. Spencer: Chalcocite Enrichment, *Economic Geology*, vol. viii, No 7, p. 622 (Oct., 1913).

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5. *The Chalcocite Enrichment*

The ores in which chalcocite is most prominent include those of the Pit and a number of levels above it.

While the capping in this area has many minerals in common with the capping which has been described, the most noticeable difference is the presence of the green and blue carbonate stains in the capping and gray ore.

a. The Carbonates in the Capping above the Chalcocite Ores.—The carbonates, together with the oxides and sulphides of copper, are present in the capping in such amounts that this capping is being deposited in large dumps where it will be treated in the future by leaching processes.

The formation of the carbonate ores may have been effected by a number of different processes, among which the most probable are: (1) The direct action of the copper sulphate solutions with lime carbonates in the rock. But the thin sections have failed to show the presence of calcite or gypsum, either or both of which should be present if this action had taken place. (2) The water running down the canyon from the limestone ridges higher up carries small amounts of $H_2Ca(CO_3)_2$ (acid calcium carbonate), which might produce carbonates, but again gypsum should be present in the ores. (3) The presence of carbon dioxide—both of atmospheric and organic origin—in the surface waters, in sufficient concentration, will produce copper carbonates when these waters come in contact with copper sulphate and sulphides. The geologic evidence shows that the latter has been the most effective process in the formation of the carbonate ores in the disseminated deposit.

Before steam-shovel operations began in 1906 the orebody was a short distance below the surface of a brush-covered mountain. It appears that during the summer months the decay of organic matter produced carbon dioxide, part of which remained in the soil; this carbon dioxide, together with atmospheric CO_2, was evidently dissolved in the water which passed down the mountain side. At the top, the carbon dioxide content was probably small, but as the water passed down the mountain side the concentration of this gas and alkaline carbonates evidently increased until, about half way down, the concentration was sufficient to produce carbonates when these waters came in contact with the accumulated copper sulphate. The atmospheric oxygen also present in these solutions was probably used up in the oxidation of the copper and iron sulphides and also in the formation of melaconite and cuprite. The sulphuric acid formed in the zone of oxidation would doubtless be partly neutralized by the surface waters. Thus the presence of the carbonates indicates that the solutions were probably slightly acid to alkaline.

b. The Formation of Chalcocite.—It is safe to say that the chalcocite in the ores below this area was formed by weakly acid to slightly alkaline

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solutions. *A*, Plate VII, shows the replacement of chalcopyrite by chalcocite, with bornite as an intermediate product. Later, the chalcocite has been replaced by melaconite and malachite. The melaconite may or may not be an intermediate product between chalcocite and malachite. In *D* it appears that chalcopyrite has been partly replaced by covellite; then a change in the solutions has caused the chalcocite to replace in part both covellite and chalcopyrite, leaving the covellite as residual crystals in the chalcocite. *C* shows chalcocite replacing pyrite, and a later replacement of chalcocite by melaconite; this replacement of pyrite by chalcocite may be a deep-seated change brought to the surface by erosion. In this specimen crystals of chalcocite shown by etching are at least ⅜ in. long and are formed by replacement of bornite, chalcopyrite, and pyrite by the copper in the descending solutions. Etching with nitric acid or potassium cyanide develops a structure almost identical with that which has been attributed to primary chalcocite; it is therefore doubtful whether etching tests can be used as criteria to distinguish primary and secondary chalcocite.

In general, the chalcocite is formed in these ores by the replacement of chalcopyrite, bornite, pyrite, and occasionally covellite. The chalcocite usually forms directly from these minerals without intermediate copper-iron or copper sulphides; however, under certain conditions chalcopyrite may be an intermediate product between pyrite and chalcocite; also bornite may be an intermediate product between chalcopyrite and chalcocite; the latter is not common, and in the same specimen the writer has not observed pyrite passing through the intermediate stages of chalcopyrite, bornite, and covellite to chalcocite, as the theoretical replacement should proceed.

In the disseminated sulphide grains the replacement by chalcocite may vary from a thin coating to complete replacement, but here, as is common throughout the orebody, the enrichment as a whole is very regular.

Whether or not the intermediate products will be formed in the replacement of iron or copper-iron sulphides by chalcocite is doubtless governed by the concentration of the ferrous sulphate in the solutions producing the reactions.

6. *Secondary Chalcopyrite and Bornite*

Secondary chalcopyrite may be formed as an intermediate product between pyrite and chalcocite; and bornite as an intermediate mineral between chalcopyrite and chalcocite, as has already been discussed, but the most interesting occurrence of these minerals is where they are formed in the reverse order by the replacement of chalcocite and covellite.

In different parts of the disseminated chalcocite and covellite ores, especially in the deeper ores, the sulphide grains appear, in the hand

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specimen, to be composed wholly of chalcopyrite; but in the polished section the chalcopyrite is often shown to be only an extremely thin coating surrounding the grain. It was probably these ores which Mr. Emmons referred to in the following quotation:[15]

"In much of the ore, especially in that of lower levels, chalcopyrite is an important ore mineral, and considerable masses . . . carry very little chalcocite. It is not known whether the chalcopyrite is a primary or secondary sulphide."

In the masses referred to as carrying very little chalcocite, it is probable that there was considerable chalcocite present, but it was covered by a chalcopyrite coating.

There are unquestionably two generations of chalcopyrite: (1) the primary, which is the most important ore mineral in the disseminated ores; (2) the secondary, which composes a minor part of the chalcopyrite. In *D*, Plate VIII, the primary and secondary chalcopyrite are shown. The primary chalcopyrite, marked *P*, forms a center surrounded by chalcocite and bornite and also one end of the grain, while the secondary forms a thin coating which in the photograph is shown as a border surrounding the grain.

The photographs in Plate VIII show the different stages in the replacement of chalcocite by chalcopyrite. In *A*, *B*, and *C* it appears that the white chalcocite has changed to a pale-blue chalcocite, due to a slight addition of iron, leaving the white chalcocite only as narrow gashes along crystallographic directions in the sulphide grain. By a gradual addition of iron the impure blue chalcocite is changed to bornite, and a further addition of iron and bornite is replaced by chalcopyrite in the form of a thin coating completely surrounding the grain, and also as small fissures and gashes extending into the grain. In *A*, *B*, and *C* the bornite and chalcopyrite are undoubtedly later than the blue and white chalcocite; the only question is the relation of the blue and white chalcocite. In the chalcocite ores which do not show the reverse reaction the white chalcocite is often formed by the replacement of the blue and it is possible that the white chalcocite in *A*, *B*, and *C* was formed by a replacement of originally blue chalcocite and is thus later; but if this has taken place there has also been a later replacement of part of the white by the blue chalcocite.

Experimental work[16] tends to show that the white chalcocite is the purest form of chalcocite, while the blue is an impure variety containing a small amount of iron; thus it might easily be formed as an intermediate product in ordinary replacements of chalcocite or in the reverse reaction where chalcocite is replaced by bornite and chalcopyrite.

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[15] W. H. Emmons: The Enrichment of Sulphide Ores, *Bulletin No. 529, U. S. Geological Survey*, p. 179 (1913).

[16] Laboratory of Physical Chemistry, Stanford University.

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The photographs *A* and *B*, Plate VI, and *A*, *B*, *C*, *D*, *E*, and *F*, Plate VIII, were selected from specimens in which all of the sulphide grains showed the reverse reactions, so these illustrations show characteristic replacements common to the specimens, rather than selected grains picked to prove a certain point.

In some of the sulphide grains it appears that a border of chalcopyrite and bornite is formed; the chalcocite assumes the pale blue color; then the chalcopyrite appears to build from the center of the grain toward the outside and from the outside toward the center. In this way chalcocite and chalcopyrite with bornite often occur as very irregular patches in the grain, but in no case has the typical graphic intergrowth of chalcocite and bornite been observed. However, the irregular formation of chalcocite with bornite and chalcopyrite often approaches a structure almost as irregular as that of the graphic intergrowth formed by eutectic mixtures of chalcocite and bornite as described by Laney[17] in the Virgilina ores.

In some grains this occurrence of chalcocite, bornite, and chalcopyrite could be explained by an incipient breakdown of the bornite into chalcocite and chalcopyrite, as described by Graton and Murdoch,[18] if it were not for the fact that in some grains (*A*, Plate VIII) the ratio of chalcocite to chalcopyrite is too large, and in other grains (*F*) the ratio of chalcopyrite to chalcocite represents a larger amount of chalcopyrite than could be accounted for by iron in bornite. Other sulphide grains show the chalcopyrite as gashes in, and bordering, sulphide grains, the centers of which are composed wholly of bornite.

Some of the sulphide grains suggest that the chalcocite in the center of the grain might have been formed by the replacement of bornite, but in grains where bornite containing gashes of chalcopyrite is being replaced by chalcocite—*C*, Plate VI—the replacement proceeds from the outside toward the center. The bornite is more readily replaced than the chalcopyrite and thus the gashes of chalcopyrite are seen extending from the bornite into the chalcocite, and often are not replaced even after the complete replacement of bornite. In the grains showing the reverse reactions this selective replacement does not occur. *C*, Plate VI, might be taken to show the appearance of a sulphide grain similar to *E*, Plate VIII, after chalcocite-forming solutions had completely replaced the chalcopyrite border and partly replaced the whole grain.

In studying the specimen from which the photographs of Plate VIII were taken, the writer has observed, in one or two grains, two generations of chalcocite, but usually it occurs in such a way as to show that there has

[17] F. B. Laney: The Relation of Bornite and Chalcocite in the Copper Ores of the Virgilina District of North Carolina and Virginia, *Economic Geology*, vol. vi, No. 4, pp. 339 to 411 (June, 1911).

[18] L. C. Graton and Joseph Murdoch: The Sulphide Ores of Copper. Some Results of Microscopic Study. *Trans.*, xlv, 49 (1913).

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been a gradual migration of the iron from the chalcopyrite border veinlets and gashes. This strongly suggests that a part of the blue chalcocite and bornite is formed as in the chalcopyrite by an addition of iron from the solutions acting on the outside of the grain.

It is interesting to find that in the covellite ores the reverse reaction also takes place and the covellite appears to be replaced directly by chalcopyrite. In some of the sulphide grains a small amount of secondary bornite is formed, but this bornite does not appear to be an intermediate mineral between the covellite and chalcopyrite. A grain of sulphide composed almost wholly of covellite is often covered by a thin coating of chalcopyrite; *A* and *B*, Plate VI, show this replacement. In *A* there is an indication of two distinct borders of chalcopyrite—marked I and II—separated by a narrow strip of covellite, which indicates that there have probably been two generations of secondary covellite and chalcopyrite. In both the covellite and chalcocite ores the chalcopyrite is formed by an addition of iron, the larger part of which has unquestionably been supplied by solutions acting on the outside of the sulphide grains.

The author believes that the secondary chalcopyrite has been formed by descending surface waters rather than the primary solutions, because chalcocite grains coated with chalcopyrite are often found near the top of the sulphide ores where enrichment extends 400 to 500 ft. below. If the chalcocite and chalcopyrite coating had both been formed by primary solutions, the thin chalcopyrite coating would undoubtedly have been replaced by secondary chalcocite or covellite. The only conclusion which can be drawn is that the local concentration of iron in the descending solutions has been responsible for the reverse reactions.

The formation of chalcopyrite by cold descending solutions has often been questioned, but recent work in the Laboratory of Physical Chemistry, Stanford University, has shown that chalcopyrite can be formed in cold solutions by the action of ferrous sulphate on chalcocite, covellite or bornite. The data in the following paragraph, together with the photographs appearing in Plate IX, are published by the courtesy of Prof. S. W. Young.

Experimental work shows that chalcopyrite can form by the direct replacement of copper in chalcocite, covellite, or bornite, by the iron of the ferrous sulphate. The reactions take place in acid, neutral, or slightly alkaline solutions in the presence of hydrogen sulphide under pressure, but are more rapid in neutral than in acid or alkaline solutions.

The photographs in Plate IV show synthetic chalcopyrite as borders and small veinlets covering and passing through fragments of chalcocite, bornite, and covellite which have been subjected to the action of ferrous sulphate for a period of about six weeks at a temperature of 30° C. A visible coating of chalcopyrite is formed on small fragments of chalcocite, covellite, or bornite in about three days; longer periods show an increase

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in the intensity of the chalcopyrite color, and polished sections made from these fragments show, under the microscope, unquestionable chalcopyrite borders and small veinlets passing through the sulphide fragment. *B* shows quartz fragments in bornite, the contact completely surrounded by narrow borders of chalcopyrite. Future work will probably show that bornite can be synthetized in a similar way.

Hydrogen sulphide probably plays the part of a catalytic agent, and its expansion from a liquid state in the sealed tube causes considerable pressure, which is probably a factor in the reverse reaction. The fact that the reaction takes place in acid, neutral, or alkaline solutions emphasizes the fact that the reactions are governed by the mass law; thus it appears that in the presence of certain concentrations of copper sulphate, covellite is formed from chalcopyrite thus:

$$CuSO_4 + CuFeS_2 = 2CuS + FeSO_4,$$

but if the concentration of the ferrous sulphate is increased beyond a certain point the reaction takes place in the reverse direction, thus:

$$FeSO_4 + 2CuS = CuFeS_2 + CuSO_4.$$

Similar equations might be written to show the reverse reaction with chalcocite and bornite.

Strong evidence supporting the reversibility of some of the equations, common to the chemistry of secondary enrichment, is given in the disseminated ores. Thus the alternating borders of covellite and chalcopyrite in the covellite ores, *A*, Plate VI, are easily explained by changes in the concentration of the iron and copper in the descending solutions. These changes might be caused in the following manner: Certain veins, pockets, or zones in an orebody are often rich in copper and others are composed almost wholly of pyrite. While the sulphides higher in copper are being oxidized the descending solutions will carry a considerable amount of copper sulphate and chalcocite, and covellite will be formed to a considerable depth; but when pyrite is undergoing oxidation the amount of ferrous sulphate will be larger and a part of the chalcocite and covellite at depth may be changed back into chalcopyrite. The change in the concentration of the iron and copper is often gradual and thus bornite replaces part of the chalcocite; then as the concentration of iron increases chalcopyrite is formed.

If the solutions are periodic, as indicated in the capping above the covellite ores, the concentration of copper sulphate and ferrous sulphate might vary considerably in different periods, and therefore it would be quite possible to have two or more generations of secondary copper and copper-iron sulphides. Where such changes take place the relation of the primary and secondary minerals will be very complex. It is very

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probable that the concentration of the ferrous sulphate must be high before the order of the replacements is reversed.

Analyses of waters from copper mines usually show the ferrous sulphate in considerable excess of the copper sulphate. Of the 18 analyses given by Emmons[19] the iron sulphate (ferric near the surface and ferrous sulphate at depth) is present in excess of the copper sulphate in all but two samples.

In any event to a certain depth the net result is enrichment, because either chalcocite or covellite is present and a considerable amount of the pyrite is replaced by chalcopyrite. *C*, Plate V, shows a pyrite grain partly replaced by chalcopyrite. The chalcopyrite appears to be formed by a replacement of chalcocite rather than by a direct replacement of the pyrite.

At present it is impossible to say to what extent ferrous sulphate has effected enrichment, but possibly some of the reverse reactions in the Butte ores, described by Sales[20] in the following quotation, were caused by ferrous sulphate rather than by primary solutions.

"Bornite . . . is often seen to be a transition product between pyrite and chalcocite; also in the reverse reaction where chalcocite passes by gradual alteration stages through bornite to chalcopyrite . . . . While bornite is commonly an alteration product, and therefore secondary, it probably does not often result from the action of downward moving surface waters, but is generally a product of ascending solutions. . . . The alteration of massive chalcocite to bornite is common, as is the change from enargite to chalcopyrite, and covellite to chalcopyrite."

Similar replacements have also been described by Graton and Murdoch,[21] and Ray,[22] but in general the primary solutions are favored. The author has not made an extensive study of the Butte ores, and as reverse reactions in them might be caused by the primary solutions as well as the secondary, the matter is open for discussion.

III. Conclusions as to the Formation of the Important Sulphides in the Disseminated Ores

Pyrite, chalcopyrite, and probably a minor amount of bornite are the primary copper minerals. The primary sulphides do not occur in these ores as magmatic sulphides, but owe their present distribution to the magnetite which was a magmatic mineral.

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[19] W. H. Emmons: The Enrichment of Sulphide Ores, *Bulletin No. 529, U. S. Geological Survey*, pp. 60 to 61 (1913).

[20] Reno H. Sales: *Economic Geology*, vol. v, No. 7, p. 682 (Oct.–Nov., 1910).

[21] L. C. Graton and Joseph Murdoch: The Sulphide Ores of Copper. Some Results of Microscopic Study. *Trans.*, xlv, pp. 44 and 50 (1913).

[22] J. C. Ray: Paragenesis of the Ore Minerals in the Butte District, Montana. *Economic Geology*, vol. ix, No. 5, p. 477 (July, 1914).

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Covellite is the most abundant secondary copper mineral forming by a replacement of chalcopyrite in slightly acid to neutral solutions.

Chalcocite is the most important secondary copper mineral formed under slightly acid to alkaline conditions.

Secondary chalcopyrite may be an intermediate mineral between pyrite and chalcocite; and bornite may be formed as an intermediate product between chalcopyrite and chalcocite.

Secondary chalcopyrite and bornite may also be formed by the replacement of the copper in chalcocite or covellite by the iron of the ferrous sulphate.

The concentration of the ferrous sulphate in the descending solutions is probably an important factor in secondary enrichment.

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Observations on Certain Types of Chalcocite and Their Characteristic Etch Patterns

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(New York Meeting, February, 1916)

IN February 1913, Prof. L. C. Graton and Dr. Joseph Murdoch[1] presented to the American Institute of Mining Engineers a notable contribution to economic geology under the title The Sulphide Ores of Copper. The value of the work was recognized by mine operators, and Mr. Graton was assisted in his plan to extend his investigations, by the organization of a commission to undertake a "secondary enrichment investigation." In this work he secured the coöperation of the Geophysical Laboratory of the Carnegie Institution of Washington. The report[2] on the chemical phases of the investigation has just been published.

Incidental to investigations[3] carried on in our laboratory of Economic

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[1] *Trans.*, vol. xlv, 26 to 81 (1913).

[2] Posnjak, Allen, and Merwin: The Sulphides of Copper, *Economic Geology*, vol. x, pp. 491 to 535 (1915).

[3] J. J. Beeson: The Disseminated Copper Ores of Bingham, Utah, *Trans.*, this volume, pp. 356 to 401.

J. C. Ray: Paragenesis of the Ore Minerals in the Butte District, Montana. I. The Covellite Zone, *Economic Geology*, vol. ix, No. 5, pp. 463 to 481 (July, 1914). II. Final paper in press (*Economic Geology*).

A. F. Rogers: Upward Secondary Sulphide Enrichment and Chalcocite Formation at Butte, Montana, *Economic Geology*, vol. viii, pp. 781 to 794 (December, 1913); (with H. W. Turner), A Geologic and Microscopic Copper Sulphide in Plumas County, California, and its Modification by Ascending Secondary Enrichment, *Economic Geology*, vol. ix, No. 4, pp. 359 to 381 (June, 1914). Secondary Sulphide Enrichment of Copper Ores with Special Reference to Microscopic Study, *Mining and Scientific Press*, vol. cix, pp. 680 to 686 (October, 1914). Sericite a Low Temperature, Hydrothermal Mineral, *Economic Geology*, vol. xi, No. 2, p. 118 (March–April, 1916). The Chemical Composition of Bornite, *Science*, New Series, vol. xlii, pp. 386 to 388 (1915).

C. F. Tolman, Jr.: Recent Advances in the Study of Sulphide Enrichment, *Mining and Scientific Press*, vol. cviii, pp. 172 to 176 (January, 1914); (with J. D. Clark), The Oxidation, Solution, and Precipitation of Copper in Electrolytic Solutions and the Dispersion and Precipitation of Copper Sulphides from Colloidal Suspensions, with a Geological Discussion, *Economic Geology*, vol. ix, No. 6, pp. 559 to 592 (September, 1914). *Microscopic Study of the Ores of the Bonanza Mine, Alaska* (in manuscript).

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Geology by advanced students and myself, and along mineralogical lines by my colleague, Prof. A. F. Rogers, we have collected data that seem to have a bearing on points raised in the papers mentioned. It was planned to delay the publication of certain of these results until further information and more definite conclusions had been obtained. However, the chemical data presented by Posnjak, Allen, and Merwin, bridge in part the gaps in our work, and the interest aroused by their research justifies the publication of a preliminary statement of the results of investigations not as yet completed.

In addition to accurate descriptions of the occurrences and structures of the copper sulphide minerals, Graton and Murdoch ventured to suggest a definite criterion to distinguish chalcocite formed from ascending solutions from that deposited by descending solutions. They believed the former to possess "a good cleavage,"[4] further defined as "three cleavages at right angles," and the latter . . . "usually shows, instead of cleavage, countless irregular cracks . . . These closely spaced cracks commonly form the outlines of individual grains . . ."[5] These structures, of course, are developed by etching. If this criterion could be established and readily applied it would have great practical value.

These suggestions of Graton and Murdoch are modified by the investigators of the Geophysical Laboratory as the result of their exact research work. They verify the suggestion of Hittorff[6] that cuprous sulphide is dimorphous. They find it to be orthorhombic below 91°C. and isometric above this temperature. Artificial chalcocite formed at high temperatures shows, on etching, an isometric cleavage, equally well developed along three planes mutually perpendicular to each other. Orthorhombic chalcocite, on the other hand, shows a basal "etch cleavage." According to my observations, the etching of that variety of orthorhombic chalcocite which is an aggregate of minute crystals develops the outlines of the irregular but roughly equi-dimensional grains, each grain having a fine lining or striation in one direction (Figs. 26 and 34). Coarsely crystalline orthorhombic chalcocite, however, is characterized by an etch pattern which is stronger and more regular in one direction than in the other directions (Figs. 38, 39 and 40).

The chemical investigations mentioned above show that both artificial and natural cuprous sulphide may contain cupric sulphide in solution, a fact also discovered by Hittorff in 1851. If more than 8 per cent. of cupric sulphide is present no inversion point from one form into the other was detected. If, therefore, chalcocite containing more than 8 per cent. of CuS is formed above 91°C. it should retain its isometric cleavage. Chalcocite from the Bonanza mine, Alaska, contains more than

[4] *Loc. cit.*, p. 54.

[5] *Loc. cit.*, p. 57.

[6] *Poggendorff's Annalen der physik und chemie*, vol. lxxxiv (1851).

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the requisite amount of CuS, and it is suggested, on account of its perfect isometric cleavage (Figs. 6 and 7), that it is "primary,"[7] and formed above 91°. My microscopic investigation does not verify this conclusion.

A discovery resulting from the chemical investigations carried out in the Geophysical Laboratory is that the blue color of chalcocite increases with the amount of cupric sulphide in solution. Of interest in this regard, I find chalcocite of pronounced blue color occurring as follows: (1) In small veinlets cutting bornite, chalcopyrite and pyrite, especially near the top of the zone of sulphide enrichment; (2) as borders around residual masses of bornite. These blue borders are evidently an intermediate product between bornite and white chalcocite and contain fragments down to sub-microscopic size of residual bornite; (3) as complicated patterns (Figs. 9, 10 and 11) in what we have designated in the laboratory as "two-colored chalcocite." The best examples of the latter were found in the ores from the Bonanza mine, Alaska. On account of its supposed isometric (high temperature) etch cleavage, its high content of cupric sulphide in solid solution, and the mooted question as to the genesis of the enormous masses of pure chalcocite outcropping at the surface, the Bonanza chalcocite is of especial interest. In this paper I will show that it has been derived from bornite (Figs. 4, 8, and 12); that the cleavage is inherited and is oriented with the two-colored patterns of the chalcocite; that the patterns are pseudomorphic after certain stages in the replacement of bornite, and therefore, in general, by these patterns we may find clues as to earlier minerals which subsequently may have been completely destroyed by the processes of chalcocitization.

I also present below examples of the control which the original bornite structure retains over the development of chalcopyrite from bornite (Figs. 19 and 20), of blue and white chalcocite from bornite (Figs. 22 and 23), and even over the development of malachite from chalcocite which in turn is secondary after bornite (Fig. 26).

Graton and Murdoch[8] recognized that bornite, breaking down along crystallographic lines, transmitted its etch pattern to chalcocite, but believed that the latter could be distinguished from true isometric cleavage of chalcocite. My studies indicate that the pseudomorphic etch pattern may develop great regularity (Fig. 27), and that a triangular or

[7] The authors appear to give the word "primary" a geological meaning; that is, they use it to designate ores formed from ascending solutions. In this paper, I use the terms "primary" and "secondary" in a strictly mineralogical sense, and therefore a statement, for example, that one sulphide is secondary after an earlier one, carries with it no hypothetical inference as to whether the ore-forming solutions were ascending or descending.

[8] *Loc. cit.*, xlv, p. 58.

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rectangular pattern (Figs. 5, 6, 7 and 8) in natural chalcocite, in which all three directions of cleavage are equally regular, proves it to be a replacement of an earlier mineral, generally of bornite. I have found the two types of etching in a single grain of chalcocite (Fig. 28), a beautiful isometric etch figure in the center where the original bornite structure controls, and a fine mosaic of striated orthorhombic crystals on the outside, where the replacement destroys instead of preserves the original bornite structure.

A type of chalcocite (Figs. 29 to 37) is figured and described below which has not been recognized by Graton and Murdoch or by the investigators of the Geophysical Laboratory. Microscopic study indicates this to be a meta-colloid. It occurs in the upper portion of the sulphide zone, and all occurrences studied so far suggest that it is formed by descending solution. It shows a characteristic and extremely minute etch structure (Fig. 34). The texture and porous character of meta-colloidal chalcocite, the manner in which it replaces pyrite, and the structure it develops upon etching are shown in photographs (Figs. 29 to 37).

To illustrate the foregoing points I have selected as far as possible ores from deposits, the geology and paragenesis of which have been carefully studied. For examples of blue and white chalcocite (Figs. 1, 9, 10 and 11) I have had to use ores from the Kennecott-Bonanza mines, Alaska. Heretofore the ore has been wrongly described as primary, and the earlier cycles of mineralization have not been recognized. This makes it necessary to summarize briefly the results of my microscopic study of this ore. The control of bornite structure through successive generations of minerals (Figs. 18 to 28) is well shown in ores from the Apache mines, Pima county, Ariz. Meta-colloidal chalcocite is figured from Bingham, Utah (Figs. 29, 30 and 31), and the disseminated deposits of Miami, Ariz. (Figs. 36 and 37). Two examples (Figs. 32, 33 and 34) from museum specimens, (locality unknown) are included, as they are the most beautiful and instructive specimens of the meta-colloidal type of ores that I have seen.

In order to make the data as complete as possible, the geological and mineralogical history of the ores is described, and in certain cases rather definite evidence is presented as to the temperature at which the minerals were formed and as to the source of the mineralizing solution, whether from below or from above.

Blue and White Chalcocite from the Bonanza Mines, Alaska

The ore deposits at the Kennecott-Bonanza mines in southeastern Alaska are unique in that they contain enormous masses of practically pure chalcocite outcropping at the surface. If we assume that these

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masses of chalcocite represent the extreme concentration of sulphide ores developed by the processes of downward enrichment, we are confronted by the fact that the oxidized zone, in which the enriching solutions had their origin, has been completely removed. Under present climatic conditions, oxidation, solution and reprecipitation are so slight that in most of the ore their results can only be recognized by the microscope, and they produce impoverishment rather than enrichment. However, the theories as to the origin of this orebody have been based largely on the belief that the chalcocite is "primary." Lindgren[9] states, ". . . there is no evidence that it" (chalcocite) "has replaced pyrite, and it is in all probability of primary origin." According to Moffit and Capps,[10] ". . . no evidence was found . . . to indicate that the orebody has ever been anything other than what it is at present. The copper sulphides appear to have been deposited as such, and a careful examination of the ore has failed to discover the presence of other minerals than those produced by the alteration of the chalcocite."

The Group of Early Minerals.—Microscopic study of ore specimens[10a] shows that there is an early generation of minerals, the remains of which are now shown by corroded residual microscopic specks. These are bornite (Fig. 4), galena, and klaprothite (?) (Fig. 1). The last two occur in minor amounts and by far the greater portion of the chalcocite is derived from bornite.

Blue and White Chalcocite of the First Generation.—That the Bonanza chalcocite is derived from bornite is proved by its intricate patterns in blue and white (Figs. 9, 10, and 11). The writer has observed that in ores from many localities, triangular patterns of blue and white chalcocite form only where bornite has suffered disintegration along its planes of cleavage in the initial stage of chalcocitization. Furthermore, examination of the Bonanza ores under high magnification was successful in discovering residual stringers of bornite (Figs. 8 and 12). These are arranged in rows, and these rows represent the last remnants of large crystals of bornite destroyed by chalcocitization. The rows, in general, are parallel to the patterns in blue and white, characteristic of the first generation chalcocite, and to the isometric etch pattern inherited from the bornite.

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[9] *Mineral Deposits*, p. 383 (1913).

[10] Geology and Mineral Resources of the Nizina District, Alaska. *Bulletin* 448, *U. S. Geological Survey*, p. 82 (1911).

[10a] The photographs of the Bonanza ores were taken during the preliminary study of a small suite of specimens presented by Theodore Chapin of the U. S. Geological Survey and Stephen Birch, President of the Kennecott Copper Corporation. Extensive collections have since been received from the U. S. Geological Survey and from Dr. J. F. Newsom, and up to date, the conclusions presented above need no serious revision.

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It appears certain, therefore, that the Bonanza chalcocite is the result of an unusually complete alteration of a great coarsely crystalline mass of bornite, and that a million tons or so of iron have been completely removed from the orebody and an equivalent amount of copper has been added by the processes that have caused this transformation.

A second pattern commonly developed (Fig. 11) seems to be pseudomorphic after the so-called "ice cake" structure. The latter is produced by the attack on the surfaces of each bornite crystal with little or no "eating in" along cleavage lines. This leaves irregular residual masses of bornite set in a matrix of chalcocite (Fig. 13).

These patterns are rarely noted on freshly polished surfaces but in short time (30 sec. in one case) the design begins to develop, and within a day or two reaches a maximum contrast, after which there often appears to be a slight fading of the color contrast. It would appear that the color is due to the unmixing of the solid solution which develops submicroscopic crystals of covellite.[11]

White Chalcocite of the Second Generation.—This appears as veinlets in the older blue and white chalcocite, and around crystal boundaries. This later generation of chalcocite is shown best by etching with potassium cyanide. In Fig. 5 are seen the large crystals of blue and white chalcocite with etch pattern inherited from the bornite, between which veinlets of white chalcocite of the second generation are injected. In Fig. 6 the parallel striations of the second generation of chalcocite contrasts strongly with the regular isometric etch pattern of the first generation of chalcocite. These later veinlets connect directly with the series of veinlets filled with malachite, and along which rosettes of melaconite are developed (Fig. 14), proving that this later chalcocitization is the result of the oxidizing solutions passing through the orebody today. Along the chalcocite veinlets, covellite needles (Figs. 15, 16, 9 and 10) "eat out" into the older chalcocite crystals. A further stage in the alteration results in the complete penetration of the large crystals of chalcocite by covellite (Fig. 15) and the transformation of the blue and white chalcocite with regular etch pattern into white chalcocite with an irregular fine etch pattern (Fig. 7). This is an example of the unmixing of the solid solution (blue chalcocite into white chalcocite and covellite) brought about by late supergene action. Chalcopyrite is developed in connection with covellite. At high magnifications specks of chalcopyrite are seen at the points of the covellite needles (Fig. 16). The

[11] The investigation of the Geophysical Laboratory proves that the blue color of the chalcocite increases with the amount of CuS in solution. It does not follow necessarily. that this is the only cause for the various colors of chalcocite. For another explanation of the blue chalcocite of the type that occurs in the complicated patterns, see article by Young and Moore cited above.

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chemistry of the process is plain. A small amount of iron remains as an impurity in the early chalcocite. It is a legacy from the bornite stage. The covellite needles grow into the chalcocite pushing the iron along in front of the crystal points until it is present in sufficient amount to force the formation of the chalcopyrite.

Summary

The paragenesis of the ore minerals of the Bonanza deposit may be summarized as follows:

First Group of Minerals.—Bornite, klaprothite (?) and galena. Temperature of formation probably relatively high for copper deposits, as Prof. Rogers[12] has found klaprothite, bornite and chalcocite to be grouped together in most of the chalcocite deposits thought to have been formed by ascending solutions, and at temperatures above those at which downward enrichment takes place.

Second Group of Minerals.—Blue and white chalcocite of the first generation, secondary after bornite. The temperatures governing, and the source of the solution causing this alteration are unknown.

Third Group of Minerals.—This includes a set of minerals forming progressively under conditions of increasing oxidation and represented as follows:

Second generation of white chalcocite $\rightarrow \left\{\begin{matrix}\text{covellite and} \\ \text{chalcopyrite}\end{matrix}\right\} \rightarrow$ tenorite (and cuprite) $\rightarrow$ malachite $\rightarrow$ azurite. This group is formed at about 0°C., as much of the ore is frozen throughout the year. It is the result of the present vadose circulation, and the copper is migrating largely as malachite (Fig. 3).

As to the structures of the Bonanza chalcocite, the two-colored patterns are portraits of earlier stages in the breakdown of bornite to chalcocite. The regular cleavage, instead of proving a primary origin of the chalcocite, indicates a secondary origin.

The original ores were coarsely crystalline bornite with accessory minerals such as galena and klaprothite (?). The process by which these have been changed almost completely into chalcocite belongs to a cycle of ore deposition which is not active at present.

The present low-temperature vadose circulation is developing the third group of minerals mentioned above, and results in an impoverishment rather than an enrichment of the ore.

The above conclusions should be taken into account in the formulation of any theory as to the origin of the first generation of chalcocite. I leave to Prof. Graton (who has had every opportunity to study the ore and the mine) the task of deciding as to whether the extreme chalcocitization that

[12] Unpublished manuscript.

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has taken place at the Bonanza mine was the result of descending solutions acting in the past under warm climatic conditions, or whether the chalcocite, undoubtedly secondary in the mineralogical sense, was the product of ascending solutions, as affirmed by Sales for the Butte ores.

Ores from the Apache Mines, Santa Catalina Mountains, Pima County, Arizona

Ores from this mine were chosen to illustrate the breakdown of bornite along crystallographic lines, and the preservation of these directions in minerals of subsequent generations. The geology and mineral paragenesis of these deposits has been worked out in detail by the writer during the mapping of the Tucson quadrangle for folio publication for the United States Geological Survey. The salient features are summarized below in order to show clearly that the first formed copper minerals, chalcopyrite and bornite, were deposited at elevated temperatures, and that the complex breakdowns and replacements described in detail are the result of alterations which are taking place at or near the surface.

Summary of the Genesis of the Ores

The deposits are of contact metamorphic origin, formed at and near the contact of a quartz diorite sill with Paleozoic limestone. The series of events which culminated in the formation of workable ores are:

I. The intrusion of a quartz diorite sill and smaller sills of albite diorite porphyry.

II. Development of contact metamorphic silicates and ores, especially in those localities where the sills contain lenses and segregations of acid pegmatitic material. This is of importance in showing the relation of the ores to the processes of magmatic differentiation. These minerals are grouped according to their order of formation as follows:

Group 1. Garnet, green pyroxene and magnetite. These are very high-temperature minerals.

Group 2. Pyrite, epidote, quartz, molybdenite, chalcopyrite (Figs. 18 and 21), bornite, all cut by tremolite. Temperature of formation moderately high, as tremolite is the latest mineral of the group, and here is an after effect of contact action. The copper minerals replace pyrite in part. Minerals of this group cut and replace minerals of the first group (Fig. 17).

III. Oxidation and enrichment of the orebody.

Group 1. Chalcopyrite (second generation), blue and white chalcocite, covellite. These are formed locally near the top of the orebody by descending solutions; ore unimportant. Temperature 0°–20°C.

Group 2. Chrysocolla and indefinite mixtures of copper oxides and carbonates forming at or near the surface. Temperatures 0°–25°C.

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Pegmatite dikes and high-temperature quartz-epidote veins (or dikes) cut the orebodies, and are believed to have been injected between the formation of groups 1 and 2 of the contact metamorphic stage. Basic dikes cut the orebody and are undoubtedly later than all the minerals of stage II. The contact minerals and ores, therefore, fall within the period of the activity of magmatic process connected with the intrusion of the quartz diorite.

The Structures of Minerals Secondary After Bornite.—Special attention is directed to the decomposition of the bornite. The alteration is undoubtedly here due to superficial solutions.

Fig. 17 shows garnet cut and replaced by chalcopyrite and bornite, and Fig. 18 the high-temperature chalcopyrite and bornite cut by, and earlier than, the tremolite needles (see also Fig. 21). The bornite in this slide (Fig. 18) has been attacked by solutions locally rich in iron and the second generation of chalcopyrite develops along the cleavage lines of the former. This type of replacement of bornite is only found in an orebody formed by replacement of a basic sill, and the source of iron is therefore at hand. Fig. 19 is a higher magnification of a much smaller speck of bornite and shows the details of this alteration. Fig. 20 shows chalcocite replacing bornite more readily than chalcopyrite leaving the latter as a residual pattern in the chalcocite. Fig. 22 presents a typical rectangular structure developed by a partial replacement of bornite by chalcocite along a veinlet, and Fig. 23 a similar regular pattern preserved in blue and white chalcocite with residual specks of bornite. Fig. 24 shows the second generation of chalcopyrite which has replaced bornite and, in turn, is attacked by chalcocite which redevelops structural lines parallel to the cleavage of bornite. Such structures were not developed by chalcocitization of the first generation of chalcopyrite. Fig. 25 shows covellite replacing bornite along cleavage lines. This is from the Engels mine, Shasta County, California, described by Rogers.[18] It is given as another example of the control of the bornite structure on later alterations. Finally, Fig. 26 represents malachite replacing chalcocite along directions inherited from bornite. This example is important in showing that preservation of structures does not involve a crystallographic orientation of the minute chalcocite crystals with the bornite structure and cleavage, and therefore, that the pseudocleavage is not true cleavage of chalcocite. The individual crystals are shown in the blue and white patterns of the chalcocite and have a fine orthorhombic striation similar to that developed by etching and described below.

As shown in Fig. 27, a regular etch pattern was developed by treating blue and white chalcocite with nitric acid. This chalcocite is similar to that shown in Fig. 23, and, of course, is pseudomorphic after bornite.

[18] *Loc. cit.*

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Fig. 28 shows both the regular etch pattern after bornite and an irregular pattern characteristic of chalcocite. The latter was developed just outside the area in which the bornite structure controls.

The control shown by the bornite structure through one or even two successive generations of minerals is merely an example of successive pseudomorphic replacement which the reflecting microscope has shown to be common in ore deposits. The best examples known to the writer of structures inherited through several generations of minerals are shown in the complicated ores of Butte, Mont. Beautiful examples are figured in J. C. Ray's final article[14] on the Butte ores.

It is a strange fact that alteration of bornite into chalcocite takes place along crystallographic planes in some cases and in other cases it attacks the crystal surfaces without regard to crystal planes and produces rounded irregular structures such as the "ice-cake" structure (Fig. 13) and the so-called "graphic intergrowths."

Summary

The ore deposits at Apache camp, Ariz., are contact deposits, the unenriched ores consisting of chalcopyrite and bornite and very minor amounts of pyrite. These are known to have been formed at high temperatures, intermediate in time between the first and last minerals produced by contact action.

The bornite especially is unstable under conditions of local oxidation and enrichment near the surface, and forms chalcopyrite (where solutions are rich in iron) blue and white chalcocite, covellite, and malachite. In the center of each speck of bornite the breakdown is always along the cleavage planes of bornite, but on the margins this structure may not control. The secondary minerals inherit the bornite cleavage as a replacement structure and not a true cleavage. Etching of secondary chalcocite re-develops the perfect isometric cleavage of bornite, especially in the center of the grains while the margins may show the more irregular cleavage or parting characteristic of chalcocite. It is suggested that a very regular isometric cleavage in natural chalcocite may be a proof of the secondary nature of the same, the antecedent mineral being bornite.

Meta-colloidal Chalcocite

As is well known, artificial precipitates of metallic sulphides are colloidal. Experiments by J. C. Clark[15] have shown that in the presence of

[14] *Economic Geology*, in press.

[15] Tolman and Clark: The Oxidation, Solution and Precipitation of Copper in Electrolytic Solutions and the Dispersion and Precipitation of Copper Sulphides from Colloidal Suspensions, with a Geological Discussion. *Economic Geology*, vol. ix, No. 6, pp. 559 to 592 (September, 1914).

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hydrogen sulphide both natural minerals and sulphide precipitates are dispersed in so finely divided a condition that they pass through pores and fine openings like true solutions. They pass through filter paper and are precipitated by the escape of hydrogen sulphide. Later experiments by Young and Moore[16] show that the sulphides of copper can be taken into solution by hydrogen sulphide and re-precipitated in crystalline form with the utmost ease.

In case the precipitate is first a gel, this may crystallize slowly, and the resultant mosaic of fine crystals may preserve the structure of the original gel. For such crystalline substances Wherry[17] introduced the term "meta-colloid."[18]

If, as suggested by Tolman and Clark, fine dispersion by hydrogen sulphide is the important factor in the transportation of metallic sulphides, and if precipitation of sulphides by the escape of hydrogen sulphide is equally important in ore genesis, the recognition of colloidal structures in sulphide minerals is of interest. Such structures might be expected near the surface and in open spaces rather than in the deep-seated ores which have undergone slow and thorough crystallization and repeated re-crystallizations by replacement. As far as known to the writer, meta-colloidal chalcocite has not heretofore been described.

Chalcocite ores from small veinlets from Bingham, collected by Prof. Rogers, showed upon examination a marked mammillary structure, typical of meta-colloidal minerals. Examination of a polished section of this ore revealed that malachite and the copper oxides were developing along conchoidal structural lines that otherwise would not have been recognizable. Figs. 29 and 30 are from this specimen and Fig. 31 shows a similar fine structure developed by etching. Later it was found that imperfect examples of this structure are common in ores developed near the top of the sulphide zone. Familiarity with this curved line and branching structure assists one to recognize this type of chalcocite by the curved and branching natural fractures, and the application of the etching test described below will settle doubtful cases. Fig. 37 shows a veinlet of this type of chalcocite replacing pyrite.

This type of chalcocite is unusually porous. Chalcocite ores in general are porous and thus ready access is given to circulating solutions of all kinds, but meta-colloidal chalcocite shows extreme development of porosity, so that it is polished with difficulty, and it is necessary to develop a thick flow surface to get a brilliant reflection. Porosity is a characteristic

[16] S. W. Young and N. P. Moore: Laboratory Studies on Secondary Sulphide Ore Enrichment, *Economic Geology*, vol. xi, No. 4 (June, 1916) and No. 6 Aug.-Sept., 1916).

[17] Variations in the Composition of Minerals: *Journal of the Washington Academy of Science*, vol. iv, pp. 111 to 114 (1915).

[18] For an excellent summary of data regarding mineral colloids and extensive bibliography see: R. Marc and A. Himmelbauer: *Fortschritte der Mineralogie, Kristollographie und Petrographie*, vol. iii, pp. 11 to 62 (1913).

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of many meta-colloids. Figs. 32 and 33 show a typical meta-colloidal chalcocite at different magnifications. Fig. 35 is a concretionary chalcocite inclosing quartz grains. The dust-like specks are the pores, the size of which can be compared with the very fine sand grains.

Chalcocite directly replacing pyrite, near the top of sulphide-enriched orebodies, especially in ores of the "disseminated type," shows extensive development of the "exploding bomb" structure due to replacement in the interior of the crystals of pyrite along crystallographic faces. This is accompanied by a development of excessive porosity in the chalcocite deposited around the crystals of pyrite, and, to a less extent, between the crystal faces, so that the outlines of the original pyrite crystals are plainly marked by cloudy chalcocite (see Fig. 36). It would appear that the first precipitate of chalcocite around the pyrite crystals and along the crystal planes opened by solution is colloidal, as is the material artificially produced with pyrite as a precipitant, and the setting (in this case, crystallization) of the gel further fractures the crystal and assists in its complete destruction.

All the above-mentioned examples of chalcocite suggestive of an original colloidal structure are characterized by an orthorhombic etch figure, consisting of aggregates of exceedingly minute crystals, each showing a typical striation in one direction. This structure is so fine that a high-power oil immersion lens is necessary to resolve it. Fig. 34 shows the etch pattern of the typical meta-colloid material figured in photographs 31, 32 and 33.

General Conclusions in Regard to Etch Figures

The following conclusions may be considered tentative to be expanded or modified as further data are collected.

No isometric etch figures have been found in natural chalcocite, that are not inherited after some antecedent mineral, generally bornite. Posnjak, Allen, and Merwin suggest that the Bonanza ore is the only natural example known of original isometric etch cleavage. I have shown this to be secondary after bornite.

Regular isometric inherited structures are found in chalcocite formed presumably by ascending solutions and certainly by descending solutions, and therefore cannot be used as criteria to determine whether the chalcocite is hypogene or supergene.

Orthorhombic etch structure with one-direction cleavage, or parting distinctly more regular than other directions, is found wherever the structure of the mineral replaced does not govern. It is found in chalcocite formed presumably by ascending solutions (example: the so-called "graphic intergrowth" from Virgilina district, Virginia (Fig. 38) and ore showing "ice cake" structure from Butte (Figs. 39 and 40) and from super-

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gene ore. It is formed where chalcocite replaces bornite or other minerals, without regard to the internal structure of the mineral replaced.

The development of a very fine orthorhombic etch structure consisting of minute individuals each lined with parallel striations (Fig. 34) is suggestive of meta-colloidal chalcocite. This is probably formed at the top of orebodies and is presumably deposited by descending solutions.[19] Good examples are found in the "concretionary" chalcocite of Butte, the rich chalcocite veinlets at Bingham, and the "disseminated ores" of Ray and Miami, etc.

All gradations exist between the very fine meta-colloidal etch structure, and coarser, more irregular structure, characterized in general, however, by one direction of fine striation for each crystal. Both of these differ from the more regular patterns, such as shown in Figs. 39 and 40. In the majority of cases the irregular type (including the fine meta-colloidal type) appears to be the result of descending solutions acting near the top of the orebody. In many cases a second generation of chalcocite has the irregular pattern and a first generation has the regular pattern.

Graton and Murdoch[20] undoubtedly had in mind these two types of pattern when they described the "primary" chalcocite as having "a good cleavage" and "secondary" as showing "countless irregular cracks." This early-recognized difference in patterns appears to be a real one, and we may conclude that the irregular pattern is formed, rather generally, nearer the surface and at lower temperatures than the regular one. On the other hand, the isometric etch pattern, pseudomorphic after bornite, is formed by both supergene and hypogene processes of chalcitization and at all temperatures at which this process takes place.

General Description of Plates

All the figures are photographs of polished sections taken with the reflecting microscope, except Fig. 17, Plate V, which is a photograph of a thin section taken with the polarizing microscope.

The magnifications, which vary from about 10 diameters to nearly 1,000 diameters, should be noted by the reader in order to appreciate the scale on which the described processes are taking place.

Plates I, II, III and IV (except Fig. 13) represent ore from the Bonanza mine, Alaska. They illustrate the occurrence of the early-formed sulphide minerals, represented by a few residual specks; the first generation of chalcocite with complex patterns in blue and white; and the second generation of white chalcocite in veinlets and the accompanying minerals, forming at present at temperatures approaching 0°C.

[19] This is the fish-scale structure of A. Perry Thompson: The Occurrence of Covellite at Butte, Mont., *Trans.*, lii, p. 574 (1915).

[20] *Loc. cit.*, p. 57.

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Plates V, VI and VII (except Fig. 25) show ores from the Apache mines, Pima County, Ariz. Contact metamorphic minerals (garnet) are cut by chalcopyrite and bornite veinlets, and are in turn cut by high-temperature tremolite needles. Hence bornite and first generation chalcopyrite were formed at high temperatures. The complex alterations of bornite are figured, which in this case take place near the surface, just in advance of oxidation. These reactions are governed by the crystallographic directions of the original bornite. The isometric etch pattern of chalcocite which is inherited from bornite is an example of the control of bornite structure on chalcocitization.

Plates VIII, IX and X illustrate the structure of colloidal chalcocite, its porosity and its etch structure. Examples of regular orthorhombic etch cleavage are figured so that comparison can be made with the etch structure of meta-colloidal chalcocite and the inherited isometric figures.

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Plate I

Fig. 1.

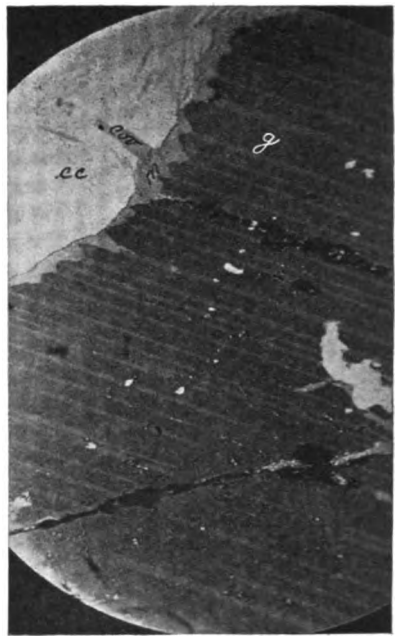

Fig. 2.

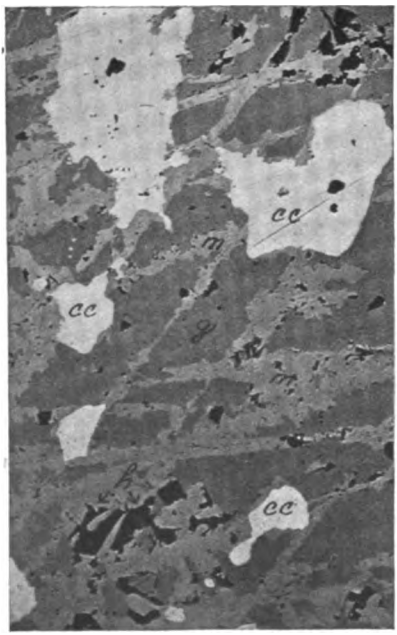

Fig. 3.

Fig. 4.

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EXPLANATION TO PLATE I

*FIG. 1.—CALCITE IRREGULARLY REPLACED BY CHALCOCITE.

Klaprothite(?) residual in chalcocite. Second generation of white chalcocite as veinlets in the main mass of first generation of chalcocite.

cc = 1st generation chalcocite.
$(wcc)_2$ = 2d generation chalcocite.
k = klaprothite(?).
m = malachite.
g = calcite gangue.
Magnification, 37 diameters.

FIG. 2.—CHALCOCITE BORDERED BY MALACHITE.

The malachite shows selective replacement of one set of the twinning bands in the calcite gangue.

cc = 1st generation chalcocite.
cov. = covellite.
g = calcite gangue.
m = malachite.
Magnification, 135 diameters.

FIG. 3.—MALACHITE VEINLETS CUTTING THE CALCITE GANGUE, AND BORDERING THE CHALCOCITE.

This relation indicates that the present cold-water circulation, acting especially along the margins of the orebodies, is carrying the copper as a carbonate.

cc = chalcocite.
m = malachite.
g = calcite gangue.
h = holes in surface.
Magnification, 63 diameters.

FIG. 4.—THE LARGEST OF THE SPECKS OF BORNITE RESIDUAL IN CHALCOCITE.

cc = 1st generation chalcocite.
dark areas = bornite.
Magnification, 136 diameters.

* Figs. 1–12 and 14–16 inclusive illustrate ore minerals from the Bonanza Mine, Alaska.

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PLATE II

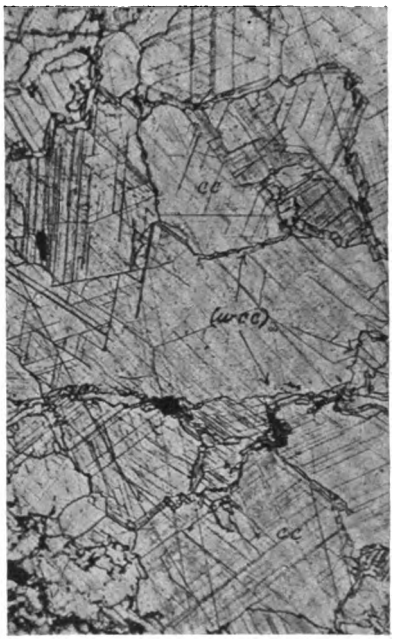

FIG. 5.

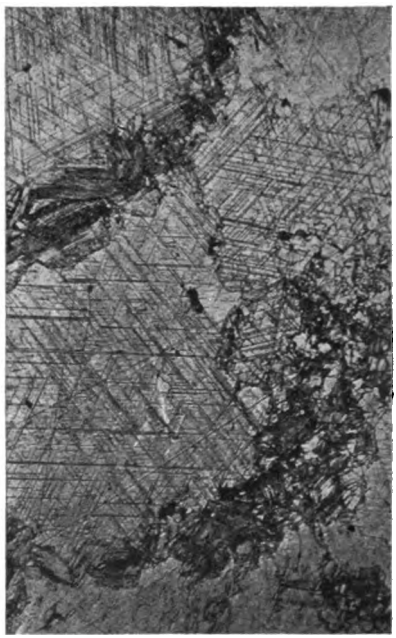

FIG. 6.

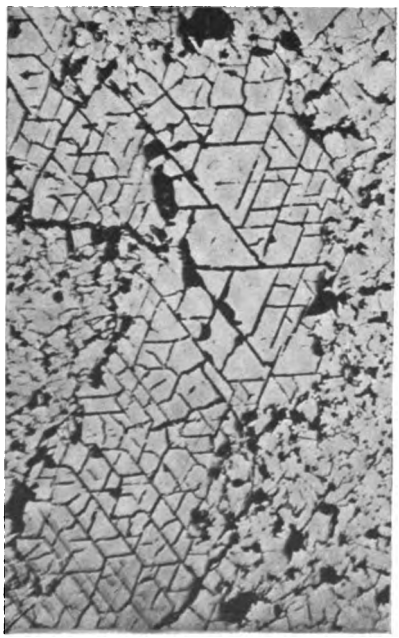

FIG. 7.

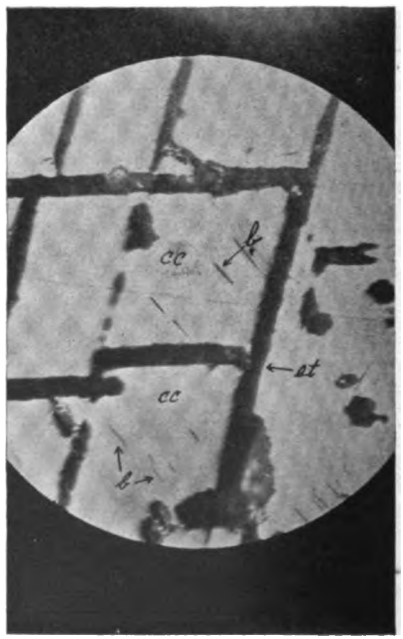

FIG. 8.

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EXPLANATION TO PLATE II

FIG. 5.—ORE ETCHED WITH POTASSIUM CYANIDE.

The crystals of chalcocite are distinguished by their regular etched patterns and are bounded by veinlets of second generation of white chalcocite accompanied by crystals of covellite (not shown in photograph—see Fig. 9).

cc = 1st generation chalcocite.
$(wcc)_2$ = 2d generation chalcocite.

Magnification, 55 diameters.

FIG. 6.—BONANZA CHALCOCITE ETCHED BY NITRIC ACID.

Chalcocite of the first generation has a regular triangular etch pattern, and the second generation chalcocite in veinlets etches in one direction only.

Magnification, 51 diameters.

FIG. 7.—BONANZA CHALCOCITE ETCHED BY POTASSIUM CYANIDE.

The large crystal of blue and white chalcocite with a regular triangular etch pattern has been altered along the margins into a white chalcocite with irregular etch pattern containing a mat of covellite needles (not shown in photograph). This is a plain case of unmixing of the first generation chalcocite into white chalcocite and covellite.

Magnification, 14 diameters.

FIG. 8.—BONANZA CHALCOCITE ETCHED BY POTASSIUM CYANIDE.

Two of the three directions of the etch pattern are shown. Stringers of residual bornite are parallel to the third etch direction which is developed elsewhere in the specimen.

cc = chalcocite.
b = bornite.
et = etch lines.

Magnification, 560 diameters.

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Plate III

Fig. 9.

Fig. 10.

Fig. 11.

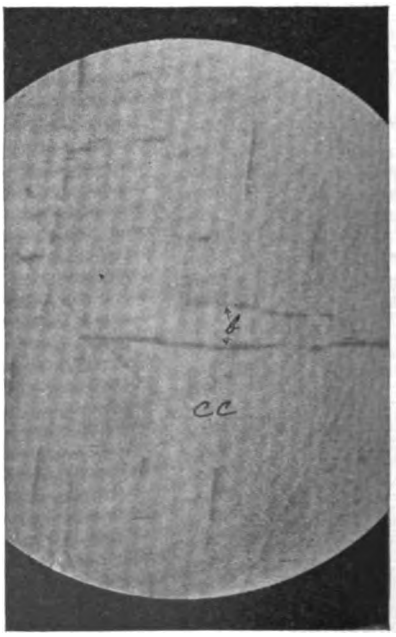

Fig. 12.

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EXPLANATION TO PLATE III

FIG. 9.—PATTERN DEVELOPED BY BLUE AND WHITE CHALCOCITE OF THE FIRST GENERATION AND VEINLETS OF WHITE CHALCOCITE OF THE SECOND GENERATION.

To the left are irregular areas of white chalcocite pseudomorphic after the "ice cake" structure developed by the irregular replacement of bornite by chalcocite (Fig. 13). On the right the white chalcocite develops along regular cleavage lines inherited from bornite. The second generation chalcocite in veinlets and borders around crystals is shown in the center of the photograph accompanied by covellite (black needles).

wcc = white chalcocite, 1st generation.
bcc = blue chalcocite, 1st generation
$(wcc)_2$ = white chalcocite of the 2d generation.

Magnification, 15 diameters.

FIG. 10.—ENLARGEMENT OF A PORTION OF THE FIELD SHOWN IN FIG. 9.

The regular cleavage pattern of white chalcocite, and covellite (black laths) along a veinlet of white chalcocite of the 2d generation are well shown.

wcc = white chalcocite, 1st generation.
bcc = blue chalcocite, 1st generation
$(wcc)_2$ = white chalcocite, 2d generation.

Magnification, 76 diameters.

FIG. 11.—A COMMON PATTERN IN BLUE AND WHITE CHALCOCITE.

The orthorhombic striation which develops on a much finer scale than the inherited "cleavage," is shown. This development resembles twinning bands.

wcc = white chalcocite, 1st generation.
bcc = blue chalcocite, 1st generation.

Magnification, 436 diameters.

FIG. 12.—RESIDUAL BORNITE STRINGERS IN THE FIRST GENERATION CHALCOCITE.

These stringers are in general parallel to the two-colored patterns and to the cleavage. They are remnants of large crystals of bornite that have been replaced along the cleavage direction of bornite.

cc = chalcocite.
b = bornite.

Magnification, 640 diameters.

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Plate IV

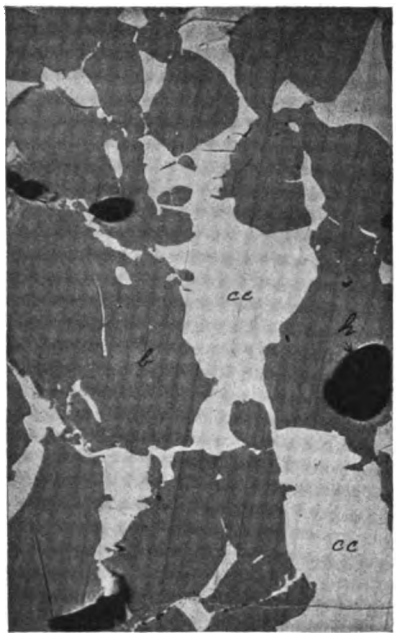

Fig. 13.

Fig. 14.

Fig. 15.

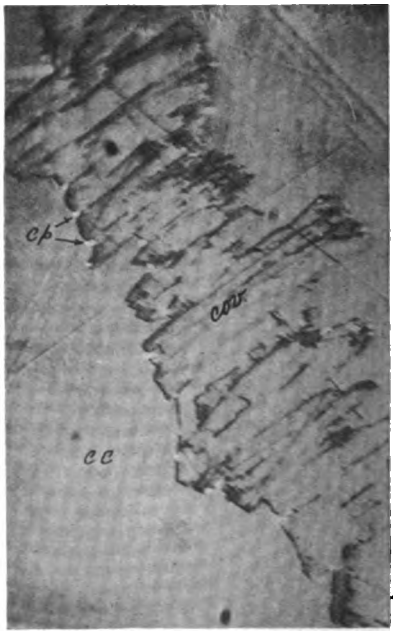

Fig. 16.

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EXPLANATION TO PLATE IV

FIG. 13.—TYPICAL "ICE CAKE" STRUCTURE DEVELOPED BY THE FORMATION OF CHALCOCITE ALONG THE MARGINS OF BORNITE CRYSTALS AND NOT ALONG THE INTERIOR CLEAVAGE PLANES.

Ore from Butte, Mont.

cc = chalcocite.
b = bornite.
h = hole.

Magnification, 74 diameters.

FIG. 14.—MALACHITE VEINLETS CUTTING CHALCOCITE AND BORDERED BY ROSETTES OF MELACONITE.

Formed where the ore is oxidizing under present conditions.

cc = chalcocite.
m = malachite.
mel = melaconite.

Magnification, 13 diameters.

FIG. 15.—COVELLITE NEEDLES DEVELOPING ALONG THE MARGIN OF CHALCOCITE CRYSTALS IN CONNECTION WITH THE SECOND GENERATION OF WHITE CHALCOCITE (NOT TO BE DISTINGUISHED IN THE PHOTOGRAPH).

cc = chalcocite.
cov = covellite.
h = hole.

Magnification, 46 diameters.

FIG. 16.—AGGREGATES OF COVELLITE NEEDLES GROWING INTO CHALCOCITE WITH CHALCOPYRITE AT THEIR POINTS.

cc = chalcocite.
cov = covellite.
cp = chalcopyrite.

Magnification, 618 diameters.

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PLATE V

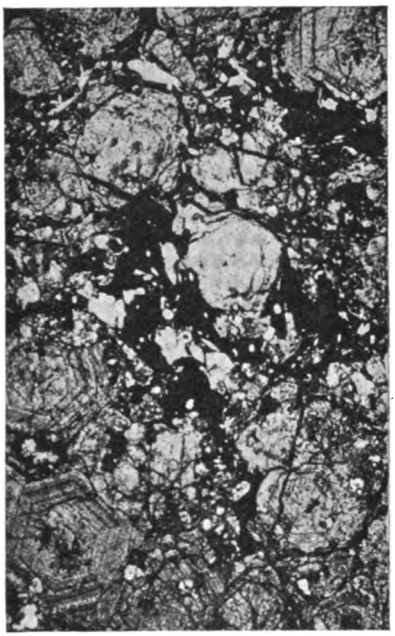

FIG. 17.

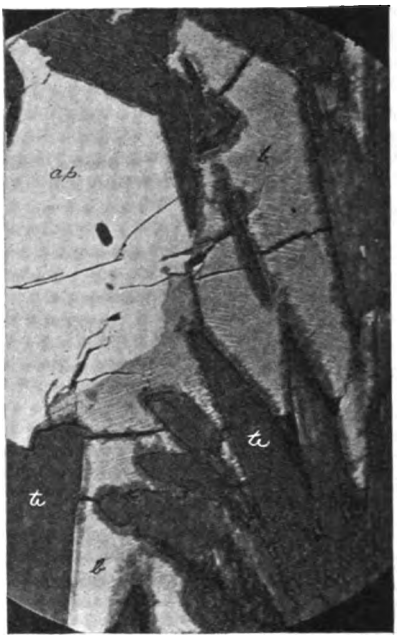

FIG. 18.

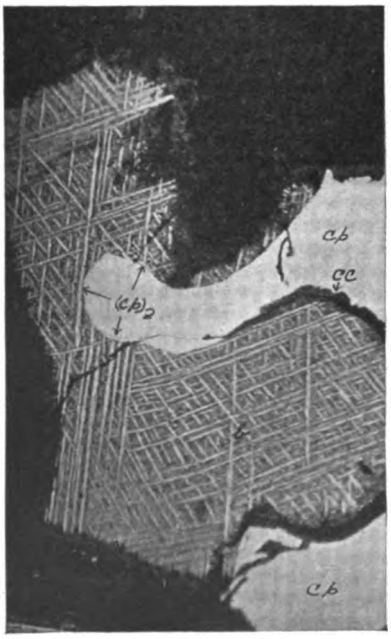

FIG. 19.

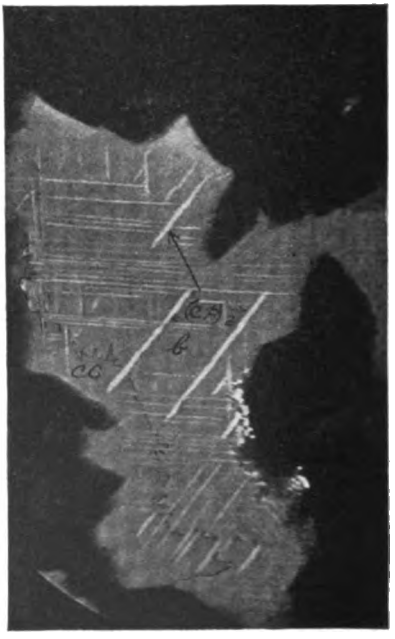

FIG. 20.

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EXPLANATION TO PLATE V

* FIG. 17.—GARNET CUT AND REPLACED BY CHALCOPYRITE AND BORNITE.

Gray is garnet. The copper sulphides are black. Thin section, upper nicol out.

Magnification, 17 diameters.

FIG. 18.—CHALCOPYRITE AND BORNITE CUT BY TREMOLITE NEEDLES.

Bornite has been attacked by supergene solutions and developed a second generation of chalcopyrite along the cleavage planes and borders of chalcocite and covellite.

cp = chalcopyrite.
b = bornite.
tr = tremolite.

Magnification, 236 diameters

FIG. 19.—HIGHER MAGNIFICATION OF A SMALL SPECK OF BORNITE AND CHALCOPYRITE TO SHOW THE SECOND GENERATION CHALCOPYRITE ALONG CLEAVAGE PLANES.

Typical hook-shaped contact developed by the high-temperature bornite and chalcopyrite is shown. Both first generation of chalcopyrite and bornite are bordered by supergene covellite and chalcocite.

cp = 1st generation of chalcopyrite.
$(cp)_2$ = 2d generation of chalcopyrite.
b = bornite.
cc = chalcocite and covellite.

Magnification, 687 diameters.

FIG. 20.—BORNITE WITH CHALCOPYRITE LINES ATTACKED BY CHALCOCITE.

Bornite is less resistant than chalcopyrite to chalcocitization. Hence residual stringers of chalcopyrite are left in the chalcocite.

cc = chalcocite.
b = bornite.
$(cp)_2$ = 2d generation of chalcopyrite.

Magnification, 720 diameters.

* Figs. 17–24 and 26–28 illustrate ore from Apache Mine, Arizona.

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PLATE VI

FIG. 21.

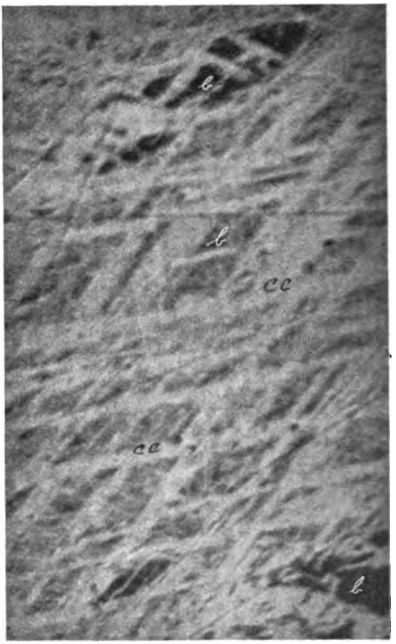

FIG. 22.

FIG. 23.

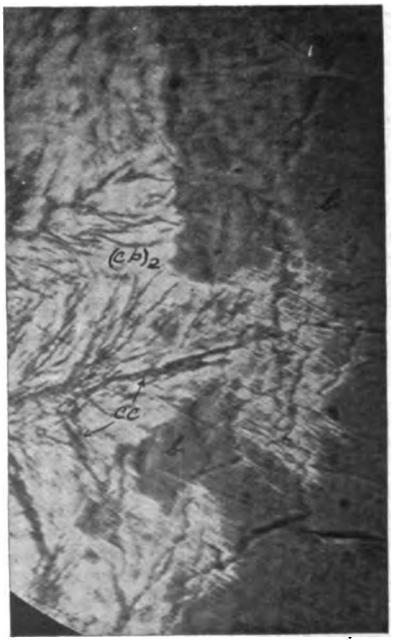

FIG. 24.

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EXPLANATION TO PLATE VI

FIG. 21.—TYPICAL REPLACEMENT OF BORNITE BY CHALCOCITE.

Higher magnifications of the same specimen show the control of bornite cleavage over chalcocitization.

cc = chalcocite.
b = bornite.
tr = tremolite.
Magnification, 55 diameters.

FIG. 22.—BORNITE BREAKING DOWN INTO CHALCOCITE ALONG ITS CLEAVAGES.

Oil was applied to surface immediately after polishing in order to prevent the development of the blue chalcocite pattern.

cc = chalcocite.
b = bornite.
Magnification, 960 diameters.

FIG. 23.—BLUE AND WHITE CHALCOCITE PATTERN PRESERVING BORNITE STRUCTURAL LINES.

A few residual specks of bornite occur in the pattern.

wcc = white chalcocite.
bcc = blue chalcocite.
b = bornite.
Magnification, 800 diameters.

FIG. 24.—CHALCOPYRITE REPLACING BORNITE ALONG STRUCTURAL LINES WITH CHALCOCITE EATING INTO CHALCOPYRITE ALONG THE SAME PLANES.

This structure was not noted in chalcopyrite of the first generation.

cc = chalcocite.
b = bornite.
cp = chalcopyrite.
Magnification, 800 diameters.

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Plate VII

Fig. 25.

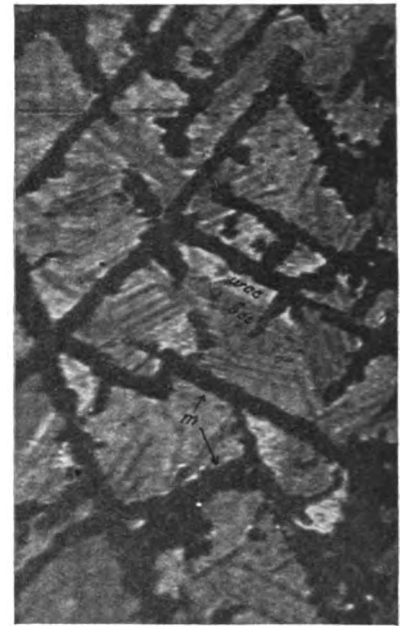

Fig. 26.

Fig. 27.

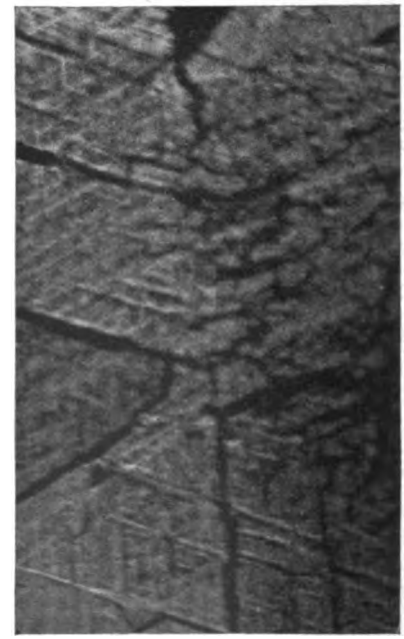

Fig. 28.

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EXPLANATION TO PLATE VII

FIG. 25.—ORE FROM ENGEL'S MINE, PLUMAS COUNTY, CALIFORNIA. BORNITE REPLACED BY COVELLITE ALONG CRYSTAL PLANES OF THE FORMER.

In this case chalcocite replaces the bornite irregularly and is not governed by the bornite structure. Photo by A. F. Rogers.

cc = chalcocite.
b = bornite.
cov = covellite.
Dark areas = silicates.
Magnification, 364 diameters.

FIG. 26.—MALACHITE REPLACING CHALCOCITE ALONG PLANES INHERITED FROM BORNITE CLEAVAGE.

The blue and white patterns in the chalcocite show that the crystals of chalcocite are not parallel to the bornite cleavages and that we have here a preservation of the structure of older minerals by replacement without crystalline pseudomorphism. This is believed to be the case generally.

bcc = blue chalcocite.
wcc = white chalcocite.
m = malachite.
Magnification, 960 diameters.

FIG. 27.—ISOMETRIC ETCH PATTERN IN CHALCOCITE INHERITED FROM BORNITE.

An area of blue and white chalcocite similar to that shown in Fig. 23, etched with HNO_3.

Magnification, 800 diameters.

FIG. 28.—MARGIN OF PATTERN SHOWN IN FIG. 27.

Outside the blue and white pattern, the inherited bornite structure does not control.

Magnification, 960 diameters.

NOTE.—Figs. 23, 24, 26, 27 and 28 are taken from one speck of chalcocite about 1/16 in. in diameter.

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PLATE VIII

FIG. 29.

FIG. 30.

FIG. 31.

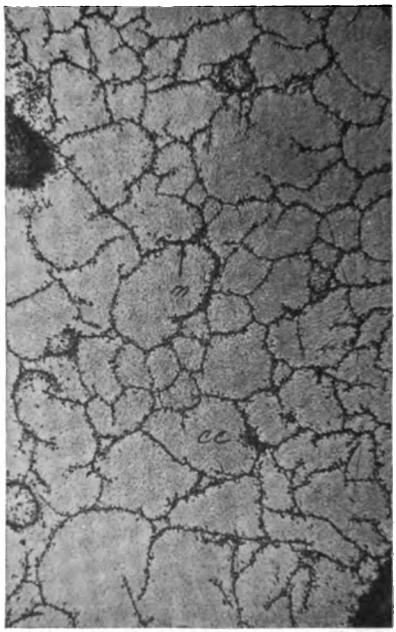

FIG. 32.

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EXPLANATION TO PLATE VIII

FIG. 29.—ORE FROM BINGHAM CANYON, UTAH, COLLECTED BY A. F. ROGERS.

Meta-colloidal concentric structure is brought out by weathering.

Light areas = chalcocite.
Darker areas = copper silicates and carbonates.

Magnification, 14 diameters.

FIG. 30.—A PORTION OF ONE RING OF THE PATTERN SHOWN IN FIG. 29.

Magnification, 44 diameters.

cc = chalcocite.
m = malachite.

FIG. 31.—ETCHED AREA ON SPECIMEN SHOWN IN FIGS. 29 AND 30.

This shows the same curved line structure in finer detail.

Magnification, 314 diameters.

FIG. 32.—META-COLLOIDAL CHALCOCITE, LOCALITY UNKNOWN.

Porosity of the chalcocite is shown in the photograph by its grayish speckled appearance. Branching veinlets are of malachite.

cc = chalcocite.
m = malachite.

Magnification, 11 diameters.

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Plate IX

Fig. 33.

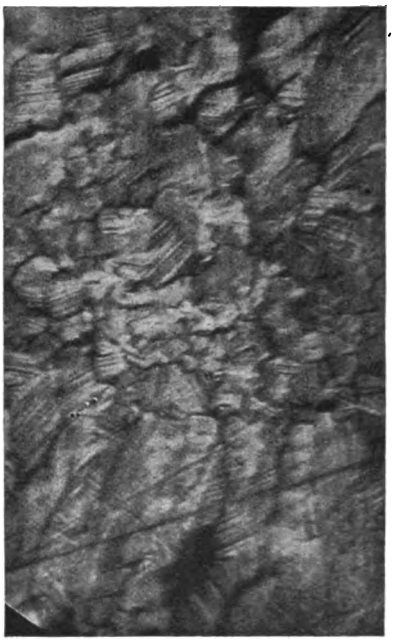

Fig. 34.

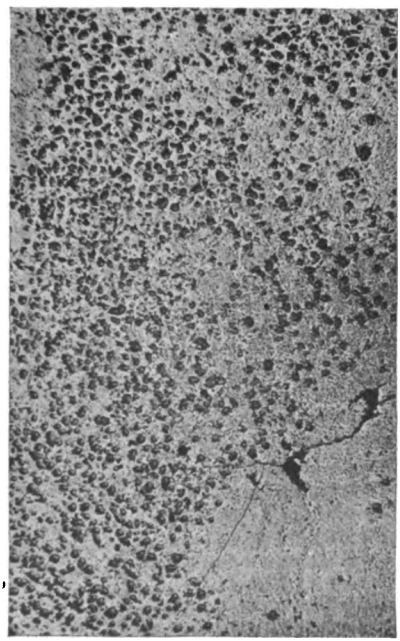

Fig. 35.

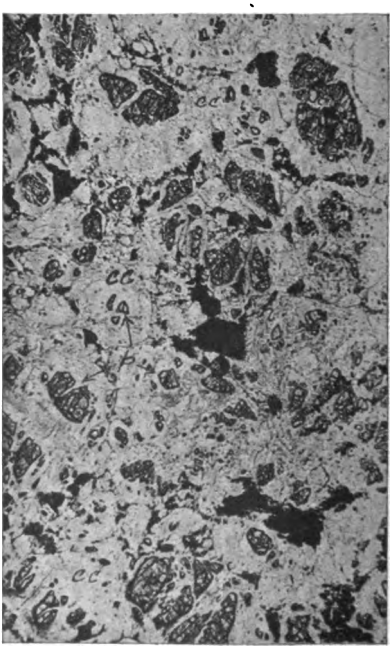

Fig. 36.

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EXPLANATION TO PLATE IX

FIG. 33.—HIGHER MAGNIFICATION OF FIG. 32, TAKEN IN ORDER TO SHOW THE POROSITY OF THIS TYPE OF CHALCOCITE.

Magnification, 93 diameters.

FIG. 34.—ETCH STRUCTURE (HNO_3) DEVELOPED IN THE SPECIMEN FIGURED IN PHOTOGRAPHS 31, 32 AND 33.

Each fine grain shows a striation in one direction. A similar pattern was developed on all chalcocites here figured as meta-colloidal.

Magnification, 960 diameters.

FIG. 35.—NODULAR ORE OF THE RED BEDS TYPE (?), LOCALITY UNKNOWN.

The chalcocite cements very fine sand and is exceedingly porous. Some idea of the size of the pores can be obtained by comparing the pattern in gray specks with the sand grains.

Magnification, 11 diameters.

FIG. 36.—"EXPLODING BOMB" STRUCTURE IN ORE FROM THE TOP OF THE SULPHIDE ZONE AT MIAMI, ARIZ.

Pyrite is forced open by the deposit of colloidal chalcocite. Streaks of porous chalcocite (dappled in photograph) outline the original pyrite crystals.

cc = chalcocite.
p = pyrite.
h = hole.

Magnification, 11 diameters.

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Plate X

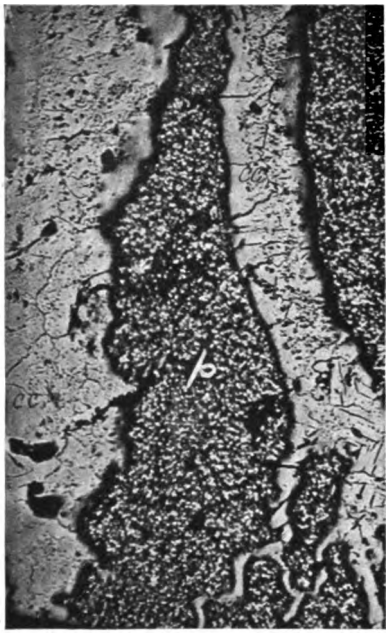

Fig. 37.

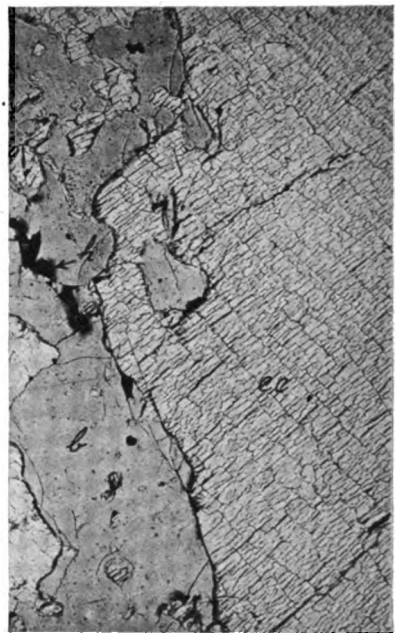

Fig. 38.

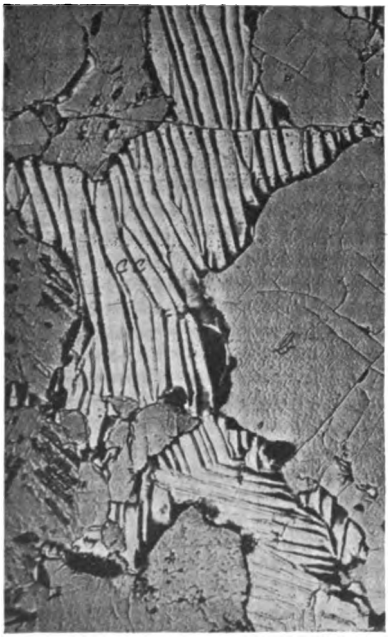

Fig. 39.

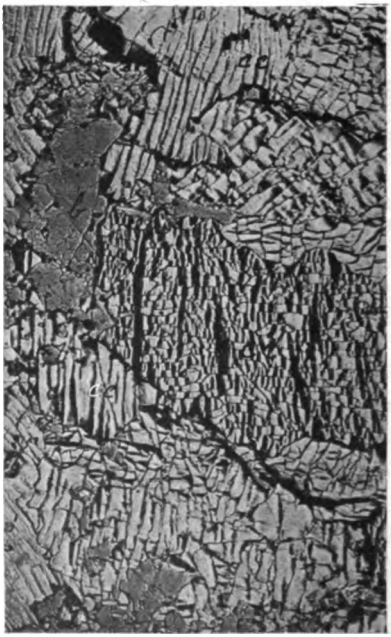

Fig. 40.

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EXPLANATION TO PLATE X

FIG. 37.—A VEINLET OF CHALCOCITE REPACING PYRITE FROM SPECIMEN SHOWN IN FIG. 36.

The vein develops an imperfect conchoidal parting and considerable porosity in the central portion (not etched).

Magnification, 59 diameters.

FIG. 38.—TYPICAL ORTHORHOMBIC ETCH CLEAVAGE IN CHALCOCITE.

This cleavage shows one direction of parting more distinct than the other direction. From Red Wing Mine, Virgilina District, Virginia. The ore shows the so-called "graphic intergrowth" (not seen in photograph) and is believed by Laney to be formed by ascending solution.

cc = chalcocite.
b = bornite.

Magnification, 56 diameters.

FIG. 39.—TYPICAL ORTHORHOMBIC ETCH CLEAVAGE IN BUTTE CHALCOCITE CONSIDERED AS HYPOGENE BY SALES, ROGERS, RAY AND THOMPSON.

From the same specimen as shown in Fig. 13. The cleavage or parting is pronounced in one direction.

cc = chalcocite.
b = bornite.

Magnification, 74 diameters.

FIG. 40.—FROM THE SAME SPECIMEN AS FIG. 39.

A parting in two directions is noted in the center of the field but is more regular in one direction than in the other.

cc = chalcocite.
b = bornite.

Magnification, 74 diameters.

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Discussion

Louis C. Graton, Cambridge, Mass.—Of particular interest at this time is any paper that deals with chalcocite, the characteristic product of secondary enrichment and the copper mineral of greatest commercial importance, and that seeks to aid in the distinction between primary and secondary chalcocite—a distinction of much scientific and practical importance but one which thus far it has proved difficult to draw. The present paper on this subject by Professor Tolman, like the many preceding it which he and his colleagues at Stanford have published in the last couple of years, contains much of interest and suggestiveness.

Without endeavoring at this time to review or comment upon Professor Tolman's conclusions in detail, I may say that many of them are in accord with the findings that my associates and I have been accumulating in the course of studying secondary enrichment in many of the copper deposits of the country. On the other hand, it goes without saying that in this comparatively new field of research or, at least, a field in which comparatively new methods are involved, absolute agreement between various sets of workers is hardly to be expected; and, as a matter of fact, with a number of Professor Tolman's conclusions I believe my associates and I are not in complete accord; with some we are certainly not in accord. As to the genesis of the remarkable ore deposits at Kennecott, Alaska, for example, with which Professor Tolman's paper largely deals, we may have something to say when the laboratory study supplementing our detailed examination of those deposits last summer shall have been made sufficiently thorough to permit us to speak with confidence about that complex occurrence.

One error which was doubtless made unintentionally ought to be pointed out. In his reference to the Secondary Enrichment Investigation and to the fact that we have enlisted in that study the coöperation of the Geophysical Laboratory of the Carnegie Institution of Washington to undertake the chemical portion of the research, Professor Tolman says: "Their report[1] on the chemical phases of the investigation has just been published." In fairness to my colleagues in Washington, I feel that so misleading an impression ought not to go uncorrected, even though the article to which he refers clearly indicates its limited scope.[2] As a matter of fact, the article mentioned, which is the second contribution thus far made by the Geophysical Laboratory to the subject of the copper sulphide ores, deals with only one of many lines of investigation which they have undertaken. Some of these investigations are now completed, so far as the work is concerned, and the results are being assembled for publication; others are in progress; still others have as yet barely been approached. Certainly the report of the Geophysical Labora-

[1] Posnjak, Allen, and Merwin: The Sulphides of Copper, *Economic Geology*, vol. x, No. 6, pp. 491 to 535 (1915).

[2] *Loc. cit.*, pp. 492 to 493.

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tory on the chemical study of the copper sulphides and secondary enrichment is not yet published. When it is published, I think it safe to assure you that it will be found a comprehensive and valuable treatment of the subject.

ALFRED C. LANE, Tufts College, Mass.—May I ask a question, Mr. Graton? In reading over that paper, I was inclined to ask this question: How do you know that those etch lines you get are always cleavage lines? Might there not be some other crystallographic faces? Do they always represent cleavage faces? Professor Tolman speaks of them as cleavage lines, doesn't he?

LOUIS C. GRATON.—In places he does, but in other places he refers to those etch lines as marking "cleavage or parting." In the minds of some who are far better crystallographers than I, there seems to be some doubt as to whether these planes of weakness are really cleavage or parting. Personally I am disposed to regard the usual etch patterns in chalcocite, for instance, as an expression of true cleavage, upon which there may be superimposed occasional planes of parting. But by "true cleavage" I do not mean to exclude the inheritance[3] by chalcocite of the cleavage of bornite from which the chalcocite has formed by alteration. Such an inherited cleavage in chalcocite Professor Tolman describes in his present paper.

J. T. SINGEWALD, JR., Baltimore, Md.—In regard to the so-called cleavage lines which are brought out in etching chalcocite: Last winter Dr. Robert Overbeck, who is now a member of the U. S. Geological Survey, made a study of the copper ores of Maryland at the Geological Laboratory of the Johns Hopkins University. These ores were formerly of some commercial importance, though at the present time they are not; they are interesting, however, for the reason that they consist chiefly of chalcocite and bornite. Dr. Overbeck found that if he took polished sections of the chalcocite which showed no indications whatever of these cleavage lines, and, without etching, placed them on the metallographic microscope, that in the course of a few moments, as the result of the heat that was focused on the specimen by the lens, cleavage lines developed, identical in character so far as the orientation and position of the lines are concerned, with the cleavage pattern brought out by etching; so that it would seem that the cleavage lines, and the lines brought out by etching, are apparently the same thing and are the actual cleavage lines in the chalcocite mineral.

CHARLES P. BERKEY, New York, N. Y.—I do not know much about the Bonanza Mine. I had just two pieces of the ore, as I recall, and I am not at all certain that my conclusions could be regarded as very well founded.

[3] Louis C. Graton and Joseph Murdoch: The Sulphide Ores of Copper, *Trans.*, vol. xlv, p. 58 (1913).

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What I want to say, however, is that after working a number of years with many groups of ores from many sources and of various sorts and with all kinds of apparatus that I knew how to use, I am becoming more and more convinced that all of these instruments are just working tools which ought to be used together. It is a great mistake, I think, to assume that polished sections of the ore, such as sulphide ores, will give you the history of the deposit. I am quite convinced that careful study of all of the associated rocks and vein matters of all sorts and especially of non-metallics, is quite as good an indication, and in some cases a good deal better, of what has happened, than the study of the metallics. In other words, this is what I want to emphasize: That a good petrographic study, with any sort of method that the mineralogist knows how to use, is the best way to get at it; and the non-metallics are even more suggestive as to origin, in most cases, than the metallics themselves.

E. POSNJAK, E. T. ALLEN, H. F. MERWIN, Washington, D. C. (communication to the Secretary*).—To prevent the perpetuation of misconception we wish to repeat our statement that artificial cuprous sulphide formed at high temperatures often[4] shows, on etching, an octahedral *parting* (not cleavage) which has *four* (not three) equally developed sets of planes making angles of 109½° (not 90°) with each other.

It is not remarkable that a structure in so many ways similar to this is found in some secondary chalcocite.

This latter structure is not defined nor consistently described by Professor Tolman; he says of it that "all three directions of cleavage are equally regular" and again "where bornite has suffered disintegration along its planes of cleavage." The first description represents cubic cleavage; about the last we are left in doubt unless we infer that the traces of cleavage (octahedral) mentioned in the textbooks is implied. There are so many "isometric" cleavages and partings that we can make little progress in utilizing such a structure until it is defined, and criteria are found for differentiating it from other structures. It may be mentioned that we have recently observed and measured octahedral *parting* in the Bonanza ore, and the color differences in this ore are as good an argument against as for Professor Tolman's hypothesis for its origin.

The terms "white chalcocite" and "blue chalcocite" should be banished at once; there is a complete series of colors. Until a quantitative color classification has been adopted, the terms lighter and darker may suggest themselves as appropriate.

C. F. TOLMAN, JR., Stanford University, Cal. (communication to the Secretary†).—It is with considerable satisfaction that I learn that Pro-

* Received Feb. 14, 1916.

[4] Melts containing little dissolved covellite may show only the etching pattern of the orthorhombic form.

† Received Mar. 21, 1916.

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fessor Graton can agree with many of the conclusions in the paper he discusses. It is to be expected that with his greater opportunity for study of some of the problems discussed, he would be able to correct errors of judgment and interpretation. It is to be regretted that he did not feel at liberty to do this, not only because the paper contains some suggestions that have not hitherto been emphasized and, if worthless, should be set aside by scientific criticism, but also and chiefly because scientific advancement is most rapid under the stimulus of frank and free discussion.

I am especially pleased at Professor Graton's insistence that careful and detailed work is essential in the interpretation of the complicated relations that this study has revealed. This should not include merely a large amount of rough examination of polished sections at the mines, important as this is in coördinating field and microscopic evidence. There is still a place for independent detailed work in a highly equipped laboratory. We have found that the answer to problems raised by intricate patterns and designs often is discovered only after the employment of the highest-power lenses available; after the reaction rims, the residual specks, and pseudomorphic patterns have been traced down to almost sub-microscopic size. Often a single polished surface has furnished material for an investigation of several weeks' duration.

In this regard it is pleasant to read Professor Berkey's remarks on the necessity of using all available tools for the investigation of ores. For determinative work, physical tests, optical tests, chemical tests, etch figures and tarnish colors all have their place, and the reflecting microscope, the polarizing microscope, and the binocular microscope, their special uses. A specimen is often sawed and the saw cut polished on one fragment and a thin section made from the second piece as close to the saw cut as possible. Often the best results are obtained by using a thin section which has received a high polish in place of a cover glass. With such a polish the "seeing" with the polarizing microscope is almost as good as with a cover glass, and observations can be made with both transmitted and reflected light. Studies have been carried on in which polished surfaces have been treated with suitable reagents and the growth and replacement of minerals have been followed under high-power lenses. These results have been especially satisfactory when hydrogen sulphide has been used. The latter is undoubtedly the great natural mineralizer causing reaction, growth, and transfer of sulphide minerals.

The results obtained in mineral synthesis[5] with the assistance of hydrogen sulphide have impressed upon me the unfortunate result of the present double meaning given the term "secondary sulphide" and I am glad that Professor Graton has given an opening for a re-statement of some of my objections.

[5] S. W. Young and N. P. Moore: Laboratory Studies on Secondary Sulphide Ore Enrichment, *Economic Geology*, vol. xi, No. 4 (June, 1916) and No. 6 (Aug.-Sept., 1916).

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It is as yet an unproved, and, I believe, an untrue hypothesis, that certain groups of sulphide minerals are to be considered as "original, initial constituents deposited simultaneously from ascending solutions" and that later changes, substitutions, and introductions are wholly the result of descending solutions and connected with the oxidation of overlying ores. It would indeed be fortunate if this were the case. The mining geologist could say then: "This mineral is later than, and produced at the expense of, an earlier mineral. It is therefore produced by "secondary enrichment" and such and such results follow." I believe that the "primary" ore minerals as well as the "secondary" minerals in most cases are introduced one after the other. Nothing could be more complicated, to use a case that has been described in our *Transactions*,[6] than the way in which the "primary" chalcocite of Butte replaces older antecedent minerals, each of which replaces an older ore or gangue mineral. As hydrogen sulphide is probably especially abundant during the early stages of mineralization, it is ever busy changing, substituting, and dissolving the unstable sulphides and forming those that are stable under the varying conditions of temperature, pressure, and chemical composition of solutions. I believe that careful study may finally establish criteria for distinguishing minerals produced by descending solutions, in spite of the fact that the chemical and physical environment of the deep (relatively) descending solutions may approach those of the near surface (relatively) ascending solutions.

It is not scientific to use a terminology that carries with it the acceptance of an unproved hypothesis. Neither is it safe to inform the miner that, if his mineral deposit is produced by ascending solutions ("primary") "there is no reason to expect that the tenor of the ore would decrease at any reasonable depth to which mining engineering has been carried." If the zone of ore deposition is the zone of escape of volatile mineralizers[7] we can understand why bonanzas of the Goldfield type are limited in depth, or at least, change in character. The limit in regard to a single constituent of an ore may be even more marked.

In every study of this kind certain facts become patent upon brief study, others are discovered after long investigation and the data for the solution of some problems perhaps are not attainable. If, therefore, we follow the scientific method of procedure of distinguishing between the proved and unproved we can talk over our results with safety even if some problems remain unsolved.

C. F. Tolman, Jr. (communication to the Secretary*).—In regard to the communication of Posnjak, Allen and Merwin it might be sug-

[6] A. P. Thompson: The Occurrence of Covellite at Butte, Mont., *Trans.*, lii, 563 to 603 (1915).

[7] Tolman and Clark, *Economic Geology*, vol. ix, p. 592 (1914).

* Received May 9, 1916.

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gested that the article is a description of the occurrence and genesis of certain types of chalcocite that may be recognized by characteristic etch patterns. The crystallographic planes that control these patterns are apparent in certain cases, but inasmuch as a complete crystallographic study of chalcocite has not yet been made, it was thought advisable to describe the patterns without reference to possible or probable cleavages, partings, twinning planes, etc.

Crystallographic directions are not easy to determine in random polished sections, as one cannot see into the opaque substance as into a thin transparent section, nor can the grains be oriented by optical tests.

It would have been of advantage to have the complete crystallographic study precede the descriptive article under discussion, but the patterns are easily recognized and characteristic, and for this reason it was thought that the description was justified. The discussion suggests that the three types of patterns described should be re-summarized for the sake of clearness.

The following types of etch patterns were noted:

(1) A pattern of great regularity which is inherited from bornite. The blue and white patterns of two-colored chalcocite in the Bonanza ore as well as the etch pattern of chalcocite of the first generation of the same ore is suggested by me to belong to this type. This type of etch pattern is common in chalcocite formed by ascending solutions (Butte) and descending solutions (Marble Peak, Arizona) wherever the structure of the antecedent bornite controls the attack of chalcocitization.

(2) A pattern of considerable regularity, but one in which one of the lines of the pattern is more regular and better defined than the others, is found in most of the Butte chalcocite, the deeper chalcocite from the Virgilina district, etc. It appears to be best developed in chalcocite formed at some distance below the surface. Where deep-seated chalcocite replaces bornite (as in the so-called graphic intergrowth) and the structure of the latter does not guide the replacement of bornite by chalcocite, this pattern is developed.

(3) The third variety includes a series of etch patterns characterized by irregular and curved lines, with a fine parallel lining defined as the "basal etch cleavage." by Posnjak, Allen and Merwin. These fine parallel lines are developed in one direction only in a single grain. Where there are two generations of chalcocite, the second generation (generally in veinlets around and through the older crystals) develops this etch pattern. In "maxillary," "conchoidal," etc., chalcocite, believed to be meta-colloidal by the writer, an extremely minute variety of this pattern is developed. Chalcocite developing this third type of etch pattern appears to be the result of descending solutions and in general is deposited rather close to the overlying oxidized zone.

As these types are illustrated in the photographs accompanying the article, little difficulty should be met in recognizing them.

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Magmatic Differentiation in Effusive Rocks

BY SIDNEY POWERS, PH. D., CAMBRIDGE, MASS., AND ALFRED C. LANE, SC. D., TUFTS COLLEGE, MASS.

(New York Meeting, February, 1916)

INTRODUCTION

THIS paper aims to present the results of an investigation concerning gravitative differentiation in lava flows, based on a quantitative microscopic and chemical study of a Triassic basalt from Nova Scotia, and confirmed by data secured from a study of certain Keweenawan ophites.

At Cape d'Or, Nova Scotia, near the head of the Bay of Fundy, marine erosion has exposed a series of Triassic basalt flows in which copper has been sought for a number of years. Studies of the thickest flow of the series with regard to volumetric composition, specific gravity, and grain have been made by the junior author, and measurements by students at Tufts College, together with confirmatory observations of the same nature on this and other flows by the senior writer, and chemical analyses made by the Division of Chemistry, Mines Branch, Ottawa,[1] all show differentiation in lava flows comparable to that in intrusive bodies. The work was made possible by the availability of the drill-cores, the property of Tufts College, and emphasizes the desirability of the study of drill-cores from other localities. Measurements of the specific gravities of certain Keweenawan lava flows of Michigan by the senior author, from material collected in 1915, will be given as a comparison to those of the Cape d'Or flow.

It has been found that the specific gravity of drill-cores, or volume per cubic foot, can be readily obtained by measuring their dimensions. This can be done to an accuracy of within 1 per cent. in pieces of core over 100 mm. in length by using a micrometer gage for the diameter, and taking the average of six measurements of the diameter; and the average of four measurements of the length made with a finely graduated ruler. Then, after weighing them, the weight in grams per cubic centimeter, or what is practically the same thing, ounces per cubic foot, is obtained. System-

[1] This paper is an abstract from a report on the copper at Cape d'Or by Alfred C. Lane with notes on the volume composition of the rocks by S. Powers, to be published in a *Report on Copper Mines and Mining in Canada*, by Dr. A. W. G. Wilson of the Department of Mines; and the chemical analyses, by M. F. Connor, are reproduced by permission of Dr. Wilson and of Dr. Eugene Haanel, Director of Mines.

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atic tests seem to show that this may be of considerable practical value, as the variation in one lava flow between the denser, more crystalline, less glassy center and the more glassy, less crystalline, and perhaps more altered upper and lower parts, can be distinctly and continuously followed when the amygdaloidal or vesicular structure is not conspicuous.

Differentiation in effusives has previously been studied by the senior writer[2] and by him and Queneau[3] the relation of size of grain to the position in the flow. The former described gravitative differentiation by fractional crystallization and gravitative separation of the feldspar to the top and of the olivine and pyroxene to the bottom in Keweenawan flows of Michigan.

Differentiation in corresponding intrusives has been described by Lewis[4] in the Palisade diabase, where the top and bottom of the sill were quickly chilled and therefore have an average composition, while differentiation took place in the remainder of the magma with the formation of an olivine-rich layer above the quickly cooled base, and a concentration of the augite below the middle of the sill, and of feldspar above the middle.

Cape d'Or Flows

Cape d'Or is a conspicuous headland projecting into Minas Channel at the entrance to Minas Basin, and rising to a height of 200 ft. above mean tide level. The sea cliff is composed of massive basalt, columnar near the shore, and is being cut back by wave attack and by tidal currents which sweep back and forth, because the range of tides is about 37 ft. at the Cape, and much more inside Minas Basin. Above the cliffs is a rolling plain upon which stands the plant of Colonial Copper Co. (now idle). On the north is an escarpment formed by the erosion of the base of the lava flows and of the underlying Triassic sandstones and shales, both of which dip south at a low angle. The escarpment faces a low plain, underlain by Triassic sandstone and Pennsylvanian sediments, that is separated from the Cobequid Mountains by a fault-line scarp. Erosion and normal faults have separated Cape d'Or from the remainder of the Triassic, so that it now has the appearance of a horst.[5]

During the course of mining operations, two shafts were sunk into the basalt, trenches were dug across the basal contact of the flows with the

[2] A. C. Lane: *Bulletin of the Geological Society of America*, vol. viii, p. 403 (1897); vol. x, p. 16 (1899); vol. xiv, pp. 369 to 384 (1903); *Michigan Geological Survey*, vol. vi, Part I, p. 106 (1898); *Michigan Geological and Biological Survey*, Pub. 6, vol. i, p. 145 (1909). See also *Trans.*, vol. xxxviii, p. 931 (1907).

[3] A. L. Queneau: *School of Mines Quarterly*, vol. xxiii, pp. 181 to 195 (1902); *American Journal of Science*, 4th Ser., vol. xiv, pp. 393 to 396 (1902).

[4] J. V. Lewis, Petrography of the Newark Igneous Rocks of New Jersey, *Annual Report for* 1907, *New Jersey Geological Survey*, p. 132.

[5] S. Powers: The Acadian Triassic, *Journal of Geology*, vol. xxiv, 1916.

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underlying sediments on the north, and seven holes drilled with diamond drills at the localities shown on the map (Fig. 1). The basalt was found to comprise five flows, with a total thickness of about 770 ft. Of this series the lowest flow is the thickest, 556 ft., and the measurements of volume, specific gravity, and grain, together with five of the chemical analyses were made from the drill-cores of it. The extrusion of this thick flow was followed in a relatively short period of time by the pouring out of three thin flows, ranging in thickness from 11 to 75 ft. With a new spurt of volcanic activity another thick flow appeared, the amygdaloidal top of which has been eroded away, leaving a thickness of only 135 ft.

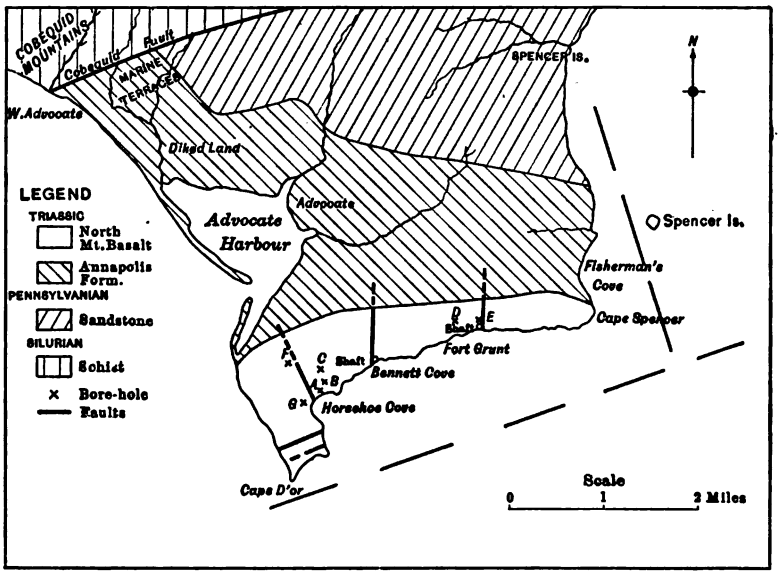

FIG. 1.—MAP OF THE CAPE D'OR REGION, NOVA SCOTIA, SHOWING THE LOCATION OF THE DRILL-HOLES *A* TO *G*.

Measurements were made on core *A*, from near Horseshoe Cove, and on core *F*, half a mile to the north. (Reproduced by permission from the *Journal of Geology*, vol. xxiv, 1916.)

Some of these flows are separated from one another by a slight amount of other material, very difficult to find on account of its paucity, which may have originally been tuff or dust as the flows spread over a land surface.

The separate flows are sometimes distinguished with difficulty both in the field, in the Triassic area in general, and in the drill-cores. Each flow is characterized by an amygdaloidal top, and a thinner amygdaloidal base. The center may or may not be vesicular: in the thicker flows it is always dense, and yet it may have vesicular streaks (as in one of the flows near Meriden, Conn.), which represent the places where the rising bubbles of gas were caught against the cooling upper portion of the flow. Usually these vesicular streaks can be distinguished from the vesicular

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or amygdaloidal top of the flow by the lack of any break and the coarse grain; but when they appear near the top, where the whole rock is vesiculated, it is very difficult to find the true top in a more or less fragmentary drill-core. Thus, in hole *F*, described below, the boundary between the two lower flows cannot be identified with certainty, and a thin flow, not represented in the other cores, may appear here. A further criterion for the top of a flow is the relative amount of alteration—greater than elsewhere—giving a lower specific gravity as shown in Fig. 4. A coarseness of grain characterizes the center of thick flows (Fig. 3), but even where this is not noticeable in the field, as in the lower flow of the First Watchung basalt near North Plainfield, N. J., a bed of tuff may separate the flows, as at that locality.

The thickness of the different flows at Cape d'Or in feet, as shown in the drill-cores, follow. Hole *D* passed through a fault-breccia and is therefore omitted. The location of the drill-holes is shown in Fig. 1.

Flow	Character (Top Eroded)	Drill-core *A*	*B*	*C*	*E*	*F*	*G*
Fort Grunt.....	Massive	103	135	132	132	81	135
		103	135	132	132	81	135
Fourth.....	Amygd.	6					
	Massive	6					
		12 115	11 146	12 144	12 144	11 92	18 153
Third.....	Amygd.	5	5			2	3
	Massive	39	39			9	38
		44 159	44 190	46 190		11 103	41 194
Goose Tongue roof....	Amygd.	20	20	18	21	13	21
	Massive	35	46	29	35	18?	44
		55 214	66 256	47 237	56 200	31? 134	65 259
Cape Spencer.....	Amygd.	20	20	19	20	26?	16
	Massive	536	(136)+	(104)+	(51)+	(520)+	(205)+
		556					
Total thickness.....		770	412+	360+	271+	680+	480+

The Fort Grunt flow is composed of massive basalt, except for the lower few feet, which shows the effects of alteration throughout, and of weathering at the upper, erosion surface. It is characterized by porphyritic labradorite (Ab_2An_3) crystals 1 to 3 mm. long; augite prisms 1 to 2 mm. long and often 0.5 mm. thick; magnetite, ilmenite, and hematite; enstatite and olivine only near the base (44 and 4 ft. from the base); and glass. The principal alteration minerals are hematite, limonite, chlorite,

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and malachite. The chemical composition is given by analyses C-13, 77, 105, 128. The corresponding thin sections show: in C-13, relatively abundant glass, alteration with the development of chlorite and limonite, although the analysis shows a high ferrous iron content; in C-77 a fresh rock with little glass; in C-105 slight alteration with more glass; and C-128, only 4 ft. from the base, a fine-grained rock with more alteration.

The Cape Spencer flow is more massive than the Fort Grunt flow, with well-developed columnar jointing. The basalt is similar to that of the latter flow, except that glass is almost lacking in the center of the flow, alteration is not so great, and the grain is coarser. The chemical analyses are C-325 (88 ft. down), A-387 (167 ft. below the top), A-492 (272 ft.), A-597 (377 ft.), A-706 (486 ft.). C-325 is a fresh basalt with abundant glass, a fine grain and some olivine; A-387 is less glassy, with more feldspar than augite, and no olivine; A-492 and 597 exhibit a comparatively coarse grain, an excess of augite over feldspar, an absence of glass and olivine; A-706 shows what is apparently a trace of micropegmatite. At A-450 (230 ft.) an inclusion of fine-grained basalt was found in the moderately coarse basalt. At A-553 (333 ft.) there are several generations of crystals, and the finer intergrowths, the last to crystallize, are probably composed of plagioclase and orthoclase with minute crystals of apatite. About 5 per cent. of the rock is composed of olivine of the same generation as the normal augite and feldspar.

The Cape Spencer flow rests on sandstone and shale of white color near the flow, but becoming red with only white streaks 10 ft. below the base. At the very base are a number of magnetite octahedra. A bed of rather soft bluish-green material also occurs at this place; it is chlorite, which may have replaced a zeolite.

Copper occurs in the flows in small quantities throughout the basalts as the analyses show, and in some of the amygdules, but principally in the veins or belts of fault breccias, especially from near the base of the Fort Grunt flow to a short distance down in the Cape Spencer flow, where it was deposited by circulating waters as native copper and as malachite.

Volumetric Composition

A series of volumetric Rosiwal measurements was made upon the Cape Spencer basalt in core *A* in order to compare the results with the chemical analyses. In Fig. 2 the results are plotted and curves are drawn to indicate the variations with depth in the quantity of feldspar, augite, and glass, with the magnetite and other iron ores included with the glass. In spite of the small number of thin sections measured, it is apparent that there is a concentration of feldspar near the top of the flow; a slightly greater percentage of augite in the center, and of feldspar near the base; a large amount of glass at both the top and bottom, but more

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at the top; a rather uniform quantity of iron ores throughout; and the presence of olivine only at the top (except the abnormal rock at 333 ft., described above, which may well be a sunken portion of the crust). It is probable that the more quickly chilled top and bottom of the flow, with almost equal amounts of feldspar and augite, represent the original composition of the magma.

In making the necessary measurements, difficulty was found by both authors in distinguishing the feldspars from a glassy rim which frequently

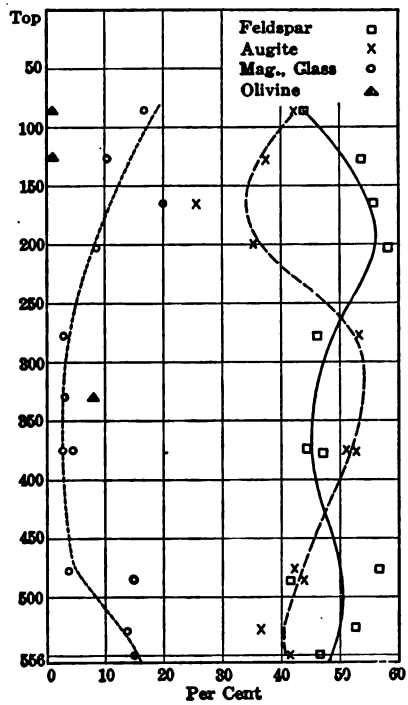

FIG. 2.—CURVES SHOWING THE VARIATIONS WITH DEPTH IN THE VOLUMETRIC COMPOSITION OF THE CAPE SPENCER BASALT, CAPE D'OR.

Feldspar is represented by the solid line, augite by the dashed line, magnetite and glass together by the dotted line.

surrounds them, especially near the base of the flow. It is best, therefore, that the curves be understood to represent tendencies, and that their qualitative value is greater than the quantitative.

The significance of the rising of the feldspar and of the settling of the augite, together with the appearance of the olivine only at the top are discussed elsewhere.

Variation in Grain

Measurements of the grain of rocks has previously been attempted by the senior author in the case of Keweenaw basaltic rocks, and by

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Queneau in the Palisade diabase, as already referred to, and the theory of grain has been discussed in detail by them.

There are several ways in which to measure the grain, and several ways in which to plot the results. The method which has been used here is to measure both the length and breadth of the largest crystals of both augite and feldspar in each slide and plot the results in areas. It seems necessary to measure both dimensions in view of the fact that certain crystals grow in different directions in different stages of develop-

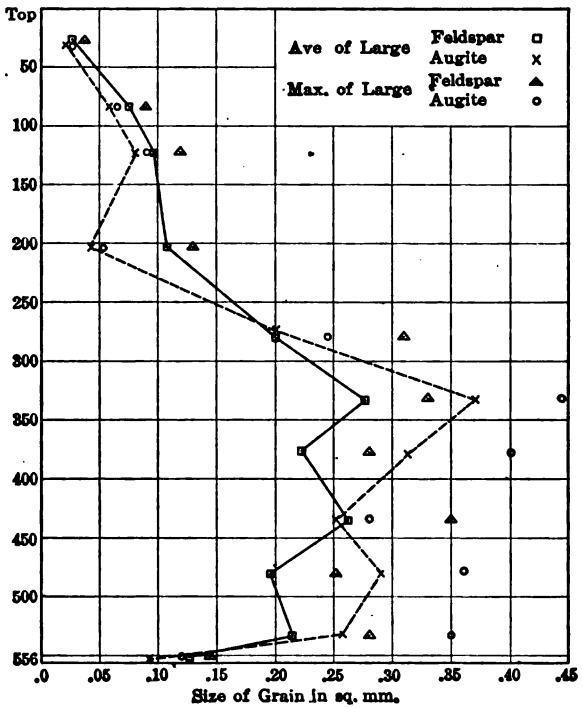

FIG. 3.—THE VARIATIONS WITH DEPTH IN THE GRAIN OF FELDSPAR AND AUGITE IN THE CAPE SPENCER BASALT.

The average area of the five largest crystals of each kind are plotted and the points connected by a continuous line in the case of the feldspars, by a dashed line in the case of the augite. The area of the maximum crystal of each kind for each slide is also plotted.

ment. Another method is to make Rosiwal measurements of the entire slide, and divide the total of the readings for a given mineral by the number of readings. This method is, however, open to the objection that several generations of crystals are usually represented in amounts varying irregularly in each slide, and that glass is not readily distinguished from some of the crystals, nor is one feldspar from another under certain conditions, especially in thin slides. The results may be plotted in terms of an average linear unit, as was done by the senior author in the Keweenaw ophites, instead of in areas, as is done here.

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In order to secure the greatest degree of accuracy, from five to fifteen of the largest crystals of both minerals in each slide were measured in order to show the existence of phenocrysts, which must not be included in the results because they represent an earlier period of crystallization. The difficulty is to select the phenocrysts, and upon the personal judgment the accuracy of this method must depend. In order to check the largest normal crystals, the average of the largest five is plotted in Fig. 3, and also the maximum (excluding phenocrysts). In actual practice it is found almost impossible to obtain the same results, quantitatively, on two measurements of the same slide. Hence, the degree of accuracy of the method is slight, and the results show merely a tendency to develop a certain grain and do not show the absolute grain. This point is further emphasized by the irregularities in the points as plotted, for curves connecting these points would cross and recross in three successive measurements.

The points show a fine grain at the top of the flow, and a coarse grain at or near the middle, with apparently a sudden transition between the two. At the base of the flow it is almost impossible to tell what is the true grain, on account of the phenocrysts, but there is apparently another sudden change to a finer grain.

Both the determinations of differentiation by volumetric measurements and by variations in grain are rather unsatisfactory, because there appears to be such a wide variation of results in any set of measurements. The methods employed are not as much the cause of this variation as is the material. The lava flow is subject to rapid chilling, and the processes of both differentiation and growth of grain are interrupted by this chilling. Also, there appear to be streaks which are either not affected by the differentiation, or else have an abnormal grain, even when they are in the center of the flow. These irregularities may be readily referred to differences in temperature and composition produced by flow currents, by engulfment of parts of the crust, and by cracks and cooling from them while the rock was yet hot and not completely crystallized. In the case of thick sills, more satisfactory results should be obtained by using the same methods of measurement.

Variation in Specific Gravity

In order to determine the relation of the specific gravity of the basalt in the Cape Spencer flow to the depth and to compare the results with those for the volume composition and grain, the specific gravities, or volumes in grams per cubic centimeter, which is the same if porosity is neglected, of 36 specimens from core *A* and 20 from core *F* were measured, the latter by students at Tufts College under the direction of the senior author. In core *A*, five of the determinations were made with a Joly

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balance; eight by the usual method of immersion with a two-arm balance; and the remainder by a method of measurement (given above on p. 442) which had previously been used by the senior author. The first and last methods were checked by the other, using the same specimens in some instances, and were found to be sufficiently accurate considering the nature of the material.

The specific gravity is affected by the porosity, but the porosity of basalts and diabases where free from amygdules and vesicles is low.[6]

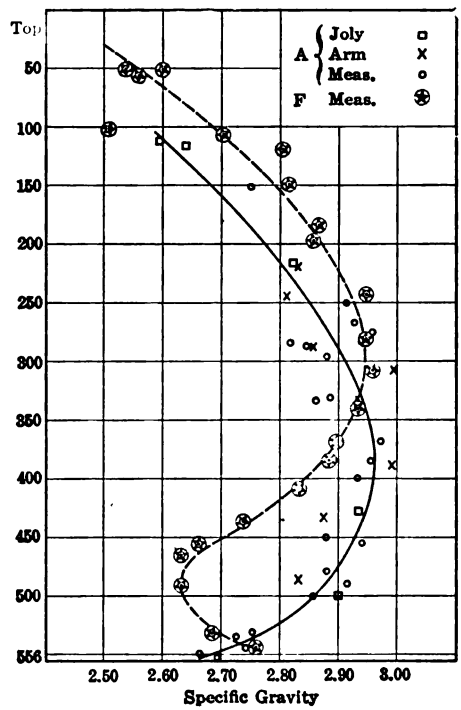

FIG. 4.—CURVES SHOWING THE VARIATION WITH DEPTH IN THE SPECIFIC GRAVITY OF THE CAPE SPENCER BASALT.

The data were secured as indicated by the use of a Joly balance, a two-arm balance, and a system of measurements. The curve for core *A* is a solid line, that for core *F* a dashed line.

If the porosity is neglected as a uniform quantity in the dense basalts, the other factors which must be guarded against are: The vesicles and amygdules; zeolitization; chloritization; and other forms of alteration, either in the rock or along veins or joints.

Curves showing the averages of the measurements of core *A* and of core *F* are shown in Fig. 4 by the solid and dashed lines, respectively.

[6] The porosity has been measured for the senior author (*Annual Report for* 1909, *Geological Survey of Michigan*, vol. i, p. 100).

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Both curves show a marked increase in specific gravity at or just below the center of the flow with a gradual decrease above and a more sudden decrease below this point. The discrepancies between the two curves are: The higher position in the curve for *F*, and the slight increase in specific gravity at the base of the curve for *F*. The curve for *F* is plotted on the assumption that the drill passed through 546 ft. of this flow. As hole *F* was bored half a mile from hole *A* and the underlying sediments were not cut, this is quite likely not exactly comparable. The irregularity at the base may also be due to local conditions at this spot in the flow. The points determined for core *A* show a great variation near the center where the basalt is entirely free from zeolitization and alteration and where long cores were available. This must be accounted for partly by streaks of heavier and lighter rock, partly by inaccuracies in measurement. At the base of the flow, chloritization and other forms of alteration give a lighter rock; at the top, zeolitization and alteration combine to give lower figures.

In order to compare the range of specific gravities along the curves, from 2.60 to 2.96, with that in other basalts the following figures are given: Kilauean basalt 2.75; tachylite 2.58; Watchung basalt 2.91 to 2.99; Mt. Holyoke basalt 2.97; and the Keweenawan figures given below.

A comparison with the results for volume composition and grain shows a marked agreement between the maximum concentration of the augite, the maximum grain and the maximum specific gravity—all slightly below the center of the flow.

The Keweenawan basaltic effusives have precisely the same differentiation, as already pointed out by the senior author, so that the hanging-wall trap of the amygdaloid lodes is easily distinguished from the foot-wall trap. Here again the upper part of the flow contains more feldspar and the lower part more augite. The upper part of the flow is then more feldspathic and tends to a porphyritic (seriate porphyritic or glomeroporphyritic) texture. This is the foot-wall trap. The lower part of the flow that contains more augite tends to a luster-mottled poikilitic ophitic texture. This is the hanging-wall trap. Most of the flows if thick enough show a trace of ophitic texture at some depth. In his 1899 paper the senior author gave analyses, not here repeated, and the specific gravity of the lower ophitic part was given as 2.877, of the upper part as 2.781. Later measurements of the weights of the drill-cores from Keweenaw Point in grams per cubic centimeter (*i.e.*, tons per cubic meter, thousands of ounces per cubic foot) show that these figures can be used as an index of differentiation precisely as in the Cape d'Or flows. Ophites give figures from 2.87 to 2.96. Feldspathic melaphyres, representing the upper feldspathic part, for instance, the foot walls of the lodes of the Quincy mine, are less heavy (2.75 to 2.78). Values below 2.75 in a rock not obviously porous generally indicate more of a felsite,

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or zeolitization. The mean value of the specific gravity of trap was found by McNair to be 2.8865 varying from 3.0904 to 2.7623. The mean value of 50 samples of amygdaloid was 2.8454 varying from 3.0936 to 2.7034.

E. Koepel, superintendent of the Champion mine, found the specific gravity of the rock tailings from that mine—running but 0.10 to 0.06 per cent. copper and with little quartz, calcite, or epidote—to be 2.84.

Chemical Composition

The chemical analyses of the Fort Grunt and of the Cape Spencer flows show closely related rocks (Bandose to Beerbachose), and the differences between the analyses of various specimens from a single flow are greater than those between the average of the two flows. The magnesia is generally lower in the Fort Grunt flow as though this flow might have been an effusion from the same source after more of the olivine had settled out.

In the Fort Grunt flow the first and last analyses represent slightly altered rocks. Below C-13 the alumina and ferric iron increase; the silica, ferrous iron, magnesia, and lime decrease. In the Cape Spencer flow there is an increase with depth from A-387 down in ferrous iron, lime, and magnesia; the concomitant changes in potash and in silica are within the limits of error, but there is a decrease in ferric iron. As the Fort Grunt analyses represent only the lower portion of the original flow, they should be compared with the base of the Cape Spencer flow. In the former there is more ferric iron, potash, and titanium, less ferrous iron, magnesia, and lime, *i.e.*, less pyroxene.

	Fort Grunt Flow				Cape Spencer Flow		
	(1) C-13	(2) C-77	(3) C-105	(4) C-128	(5) C-325	(6) A-387	(7) A-492
Distance from base of flow, ft.	119.0	55.0	27.0	4.0	437.0	389.0	284.0
SiO_2	50.14	54.04	52.98	52.30	52.50	53.00	51.92
Al_2O_3	14.56	13.82	14.08	14.86	14.30	14.71	13.25
Fe_2O_3	4.40	3.25	4.02	5.38	6.15	7.30	2.28
FeO	7.28	6.95	6.34	5.92	5.14	3.32	7.16
MgO	5.46	6.34	5.44	5.55	6.08	4.67	9.05
CaO	10.10	10.12	9.72	9.80	9.08	9.40	11.22
Na_2O	2.59	2.51	2.93	2.17	2.56	4.11	2.42
K_2O	0.55	1.20	0.96	1.36	0.66	0.71	0.64
H_2O	2.80	0.80	2.00	0.84	1.64	1.00	0.65
TiO_2	1.30	1.06	1.20	1.18	1.24	1.30	0.78
P_2O_5	0.15	0.16	0.14	0.16	0.14	.....	.....
S	0.01	.....	0.01	.....	.....	.....	.....
CuO	0.026	0.011	0.003	0.005	0.002	.....	.....
MnO	0.15	0.15	0.15	0.20	0.18	0.13	0.14
BaO	trace	trace	trace	trace	trace	trace	trace
	99.51	100.23	99.97	99.72	99.67	99.65	99.51

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	Cape Spencer		First Watchung Sheet			Connecticut	
	(8) A-597	(9) A-706	(10)	(11)	(12)	(13)	(14)
Distance from base of flow, ft.	179.0	70.0					
SiO_2	51.76	51.56	51.77	51.09	50.19	52.37	52.40
Al_2O_3	13.49	13.57	14.59	14.23	14.65	15.06	13.55
Fe_2O_3	1.71	2.00	3.62	2.56	3.41	2.34	2.73
FeO	7.42	7.80	6.90	7.74	6.96	9.82	9.79
MgO	9.05	8.26	7.18	7.56	7.95	5.38	5.53
CaO	11.62	10.80	7.79	10.35	9.33	7.33	10.01
Na_2O	2.38	2.58	3.92	1.92	2.64	4.04	2.32
K_2O	0.67	0.69	0.64	0.42	0.75	0.92	0.40
H_2O	0.60	0.90	2.31	2.67	3.04	2.24	1.67
TiO_2	0.79	1.14	1.13	1.30	1.13	0.21	1.08
P_2O_5	.....	.....	0.18	0.16	0.18	.....	0.12
MnO	0.12	0.12	0.05	0.25	0.07	0.32	0.26
Remainder	.....	.....	.....	.....	.....	.....	0.13
	99.61	99.42	100.08	100.25	100.30	100.03	99.99

Analyses 1 to 9 by M. F. Connor; 10 to 12 by R. B. Gage (*New Jersey Geological Survey, Annual Report for* 1907, p. 148); 13 by J. H. Pratt (*Bulletin No.* 6, *Connecticut Geological Survey*, p. 184 (1906)); 14 by W. F. Hillebrand (*Annual Report, U. S. Geological Survey*, vol. xxi; Part 3, p. 77 (1901)).

1 to 4 Fort Grunt flow at Cape d'Or. In core *C* the thickness of the flow is 132 ft. The top is everywhere eroded away.

5 to 9. Cape Spencer flow at Cape d'Or 5 is 88 ft. below the top of the flow in core *C*; 6, 167 ft. below the top in core *A*; 7, 272 ft., in *A*; 8, 377 ft., in *A*; 9, 486 ft., in *A*, and 70 ft. from the base.

10 to 12. First Watchung sheet in Hartshorn's quarry, near Springfield, N. J. 10 represents the "upper gray layer;" 11, the "middle black layer;" 12, the "lower gray layer."

13. Main sheet basalt, Meriden, Conn.

14. Main flow of basalt, Pine Hill, South Britain (Pomperaug), Conn.

In the Watchung analyses (10 to 12), from the top downward there is an increase in magnesia and lime, a decrease in silica, and toward the center an increase in lime and a decrease in potash and ferric iron. An increase with depth in lime and a decrease in soda were found in the Keweenaw rocks.

The combined result shows that there is an increase downward in the percentage of ferrous iron, magnesia, lime, and perhaps potash; a decrease in soda. The change in the percentage of silica is slight. Comparing the chemical composition of the minerals, which have been shown by the microscopic measurements to vary in amounts in different parts of the flow, with the data derived from an examination of the chemical analyses, a close agreement is seen. A separation of feldspar, with a composition of

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4.6 per cent. Na_2O, 12.4 per cent. CaO, 20 per cent. Al_2O_3, 59.2 per cent. SiO_2, to the top; as against olivine, with 57 per cent. MgO and 43 per cent. SiO_2, or pyroxene, with 6 per cent. Fe_2O_3, 18 per cent. FeO, 13 per cent. MgO, 11 per cent. CaO, 3 to 5 per cent. Al_2O_3, 48 per cent. SiO_2, toward the bottom, must cause the MgO and Na_2O to vary more than the alumina or silica.

The olivine appears in the thin sections mainly near the top of the flow, although in the Palisade sheet there is, as shown by Lewis, a concentration at the base. Furthermore, the olivine tends to sink in a flow just as in an intrusive body. The discrepancy is explained by the work of Bowen and Andersen[7] as a result of their recent experiments which show that olivine that has crystallized out above 1,557° may be partly or wholly resorbed during the normal course of crystallization simply as the result of cooling, but that this resorption may fail to take place if the cooling is very rapid as it would be near the top of a flow. Also Bowen has shown[8] that differentiation in silicate liquids may take place by the rising and sinking of crystals. His experiments show that both olivine and pyroxene tend to sink, and that with an excess of silica pyroxene crystallizes first. After the crystallization of the artificially prepared liquid, the upper part consists of pyroxene and free silica, the lower part of pyroxene and olivine. In the basalt flows there is correspondingly more silica at the top of the flow than at the base. It will be remembered that at a range of from 1,890° down to 1,387° Bowen found forsterite (olivine) to crystallize out, but that at about 1,557° it changed to clinoenstatite (pyroxene).

It may be suggested as a possible reason for the slight accumulation of olivine as compared with the Palisade intrusion, that the latter came to rest when at a higher temperature and lingered at a temperature corresponding to the inversion temperature found by Bowen much longer than did the flow so that olivine could form and could settle out, while in the flow the motion and consequent stirring continued until the whole had reached a lower temperature, but then remained rather long at temperatures at which the particular olivine present inverted to pyroxene and could sink as pyroxene.

This supposition is supported by two additional lines of argument. First, a sinking of pyroxene would make much less difference than a sinking of olivine in a rock which, as the margins show, had originally about 50 per cent. silica. Secondly, the broad zone of relatively uniform

[7] N. L. Bowen and O. Andersen: The Binary System MgO-SiO_2, *American Journal of Science*, vol. xxxvii, p. 499 (1914); N. L. Bowen: The Ternary System; Diopside-forsterite-silica, *Idem*, vol. xxxviii, p. 264 (1914); O. Andersen: The System Anorthite-forsterite-silica, *Idem*, vol. xxxix, p. 453 (1915).

[8] Crystallization-Differentiation in Silicate Liquids, *Idem*, vol. xxxix, pp. 175 to 191 (1915).

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coarsest grain in the Palisade diabase compared with the narrow zone at Cape d'Or (Fig. 3) indicates clearly that the former was much farther above consolidation temperature than when it came to rest, as has been shown elsewhere by the senior author.

Conclusions

The nature of gravitative differentiation in extrusive rocks has been investigated with the aid of volumetric and chemical analyses, and determinations of specific gravity and grain. The results are in accord and show a concentration of the leucocratic, felsic constituents at the top of the flow; the melanocratic, mafic constituents at the base. The quickly chilled top and bottom of the flow show approximately, when free from alteration, the original composition of the magma.

The grain of the rock in thick flows varies with the depth and is greatest just below the center, where cooling takes place slowest. The grain is difficult to determine in many cases on account of the presence of several generations of crystals, due to convection currents and the subsidence of partially cooled upper portions of the flow; in other cases, as at the top and bottom, on account of the presence of phenocrysts with a rather glassy groundmass.

The specific gravity determinations have been made in large part by a simple and rapid system of measurements in the case of drill-cores of uniform diameter and a length of over 10 cm. This method has been checked by determinations in the usual manner, and found to be of sufficient accuracy to be of much practical use.

The results of the investigation of differentiation are found to agree with those in experiments recently made by Bowen and Andersen on artificial solutions; that heavier minerals, as olivine and pyroxene, tend to sink while very light minerals may rise.

Discussion

N. L. Bowen, Washington, D. C. (communication to the Secretary*). —This detailed study and complete demonstration of differentiation as a result of the sinking of crystals has a great general significance in petrogenesis. If this action can be demonstrated in an effusive body, even though a very thick one, how can one doubt its possibility in large intrusive bodies?

The authors speak of the sinking of augite and the rising of plagioclase but I should consider that they should depend entirely on the former action to obtain their results. It is true that in my experiments, to which the authors refer, a floating of crystals was in one case obtained, *viz.*,

* Received Feb. 11, 1916.

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in the case of tridymite crystals, but it must be remembered that tridymite has an exceptionally low density. Plagioclase crystals, on the other hand, are of nearly the same density as basaltic liquid. Under magmatic conditions they might be a little heavier than the liquid or a little lighter, we do not know which, but the margin would be exceedingly small and any motion resulting, whether upward or downward, would be correspondingly small in the usual case. For this reason I should think that the sinking of pyroxene should be considered probably the sole factor in producing the observed result, for which purpose it appears to be entirely adequate.

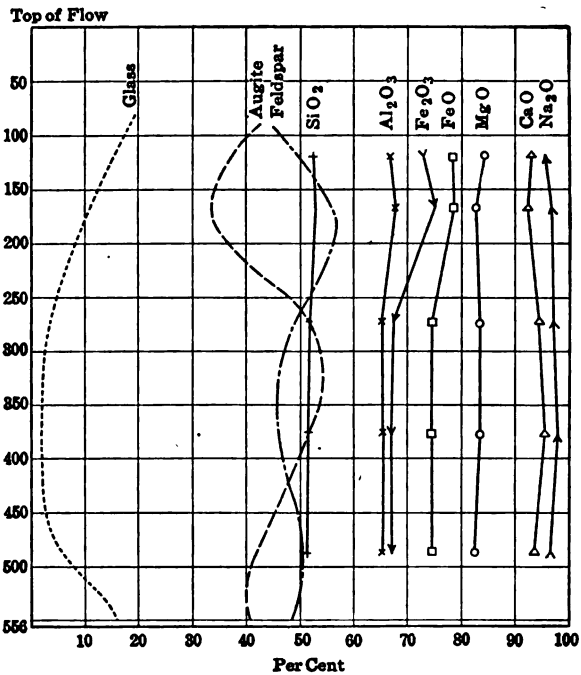

FIG. 1.—CURVE SHOWING THE VARIATIONS WITH DEPTH IN THE VOLUMETRIC COMPOSITION OF THE CAPE SPENCER BASALT, CAPE D'OR, COMPARED WITH THE COMPOSITION IN OXIDE.

The measurements of grain have also a certain significance in this connection. In the experiments referred to a difference of grain was obtained in different parts of the charge, which was entirely independent of the rate of cooling but connected with the fact that a sinking crystal does not depend entirely upon diffusion for the collection of its material from the surrounding liquid. The crystals which sank a considerable distance were therefore exceptionally large. Now in the Cape Spencer flow the augite crystals are normally (top and bottom) somewhat smaller than the plagioclase crystals, but in the layer somewhat below the middle,

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in which considerable accumulation of augite has been demonstrated, the augite crystals are there considerably larger than the plagioclases. It would seem highly probable that the exceptional size of the augites of this layer was connected with the fact that they moved (sank) through the liquid.

In any case where crystals move as they grow, Lane's theory of grain should be applied with some caution since the theory connects the size of a crystal with the rate of cooling at various distances from the margin and must of necessity assume that a crystal found at a certain distance from the cooling margin experienced its entire growth in that position. The motion of a crystal introduces another variable which is difficult to evaluate for any particular case, and which detracts somewhat from the value of measurements of grain as a means of determining the "superheat" of a magma, *i.e.*, the excess of the temperature of the magma as intruded over its temperature of crystallization. In the foregoing no criticism is intended of the general value of measurements of grain nor of their usefulness in determining the position of amygdaloidal margins.

ALFRED C. LANE and SIDNEY POWERS, Tufts College, Mass. (communication to the Secretary*).—It is quite true, as suggested by Bowen, that the settling of certain constituents should affect their grain. To it may well be due the seriate and wide variation in the size of the grains at times. As to the suggestion that the olivine in one section at 333 ft. down (A. 553) is due to a general settling of olivine and is comparable to the palisadose of the Palisade intrusive, this seems to us unlikely. Other sections between it and the top and at about the same level show no olivine, and this section is abnormal in other respects.

At F. E. Wright's suggestion we present Fig. 1 showing side by side the variation of oxides and of the augite and feldspar. The nearly constant amount of alumina shows that the later formed augite contained more alumina than the earlier. The increase of soda is not proportional to the increase of feldspar, and it seems that the oxidation of the iron may aid in sending the soda and lime into the feldspar rather than the augite.

* Received Mar. 20, 1916.

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Conservation and Economic Theory*

BY RICHARD T. ELY,† PH.D., LL.D., MADISON, WIS.

(New York Meeting, February, 1916)

I. Conservation Defined and Described

Conservation Means Preservation, Improvement, Justice

Conservation, narrowly and strictly considered, means the preservation in unimpaired efficiency of the resources of the earth; or in a condition so nearly unimpaired as the nature of the case, or wise exhaustion, admits. And broadly considered, it means more than the word itself implies, for it naturally includes an examination of methods whereby the natural inheritance of the human race may be improved; and still more broadly considered—and as used in popular discussion—it includes a treatment of the effects of productive conservation measures upon distribution. We shall give our attention briefly to the main points in this informal definition before we pass on to the thesis that conservation is largely a matter of property relations—that a wise conservation policy means wise property relations.

Preservation of Natural Resources

First, conservation suggests simply preservation, and, in the treatment of the subject which we have had in the United States, emphasis has been laid upon the past and present waste of natural resources and upon the means of putting a stop to this waste. The forests have been depleted; soil has been washed by rapidly flowing surface water from mountain sides. These rapidly flowing waters have produced high streams and devastating inundations, followed by low streams and impeded navigation. The forests have, moreover, been so removed as to bring great danger of fire, and destructive conflagrations have resulted in large waste of resources, including in some cases a destruction of the humus, supplying essential elements in the fertile soil. As a remedy we must have measures to reforest mountain sides, and this means large social control, very frequently taking the form of public ownership and management; but in addition we always find actual and proposed

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measures of coöperation with private owners. Illustrations are afforded by the services of publicly employed foresters in giving advice and in making plans, etc. Here the main emphasis is on cessation of waste and restoration of natural advantages. This means a great deal and involves hard tasks. It is said that there are places in southern Europe where it will take 300 years to restore the soil washed from the mountain sides, as it is necessary to begin at the bottom and by planting trees gradually to build up a fertile soil from falling leaves, decaying vegetation, etc.

Another feature of the existing situation much discussed is the enormous waste of good soil by the erosion of water, even in the more level portions of the earth's surface, as in the Mississippi Valley, where millions upon millions of tons of previous elements of fertility are being annually wasted—washed into this turbulent stream. It has been proposed to construct at public expense mighty reservoirs to check the rapidity of flow and to hold back the soil, which would also facilitate navigation. Here is one suggestion, and without entering into the extent of its possibilities, it is again apparent that emphasis is laid on preservation as nearly unimpaired as possible, as careful an exhaustion of supply as the nature of the case admits.

Improvement of Natural Inheritance

But, in the second place, we are concerned with measures to improve the natural inheritance of the race, and this is brought out more or less fully in discussions of conservation. Under a rational system of forestry it is claimed that an acre of ground will produce two or three times as much in a given period of time as the same acre in a wild state of nature, when the trees are allowed to grow up naturally without man's care. Conservation of water adds to the fertility of the soil or at any rate brings about a better utilization of the fertile elements when the water is applied to the land in irrigation. Modern agriculture, which is at least partially included in discussions of conservation, is not content with the maintenance of an existing status, but wants to improve what is. Nearly everything needed to accomplish this purpose may be found in the land itself, or placed there by cultivating green crops and plowing them under, and perhaps extracting from the air elements of fertility to be united with the soil. The phosphates and potassium seem to be the main, if not the only, elements which must be added from without and which are in no physical proximity to the land, generally speaking. It must be admitted that the supplies of these are far more limited than we could wish, although it is highly probable that now unknown sources of supply will be discovered. Certainly there is no ground for present alarm, and for this assurance there are several reasons. With a general

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appreciation of the natural scarcity of these elements, economy in their use will be practised to a greater extent than previously, and this economy will be further promoted by an increase in price, accompanying growing scarcity. The natural result will be to establish something approximating a cycle, whereby these elements can be returned to the land again and again. We see something of this sort in countries like China and Japan. We are, therefore, warranted in the belief that for an indefinite time to come, certainly for several centuries, the productivity of the land may be maintained and even increased.

Conservation Concerned Primarily with Production

In the third place, we observe that conservation considered as a part of economics has to do first of all with that division of economics which we call production; not with production in the technical sense, but in the social sense; in other words, with relationships arising out of the efforts of men, associated to produce wealth. The question here has to be asked what institutions or what shaping of institutions is most favorable to conservation? Here, as elsewhere in economics, we have to do with property and contract, and find we cannot make much progress until we have adopted the social theory of property and the social theory of contract.[1]

Justice in Distribution

And, in the fourth place, conservation takes cognizance of distribution and aims to bring about justice, as the conservationists see it. In general, it may be said that the conservationists wish to cut off, or at least reduce, the private receipt of property and income beyond what is a fair return to capital and labor and enterprise, reserving the surplus for public use.

Notice that we now get beyond the limits of conservation, strictly speaking, and are quite within the field of economics, but it would be arbitrary to stop the discussion at this point, where it yields fruitful results. This simply shows that conservation is in large part economics, one of our fundamental theses.

Conservationists seek to control franchises for the use of waterpower with this end in view, exacting payments for the public treasury in cases where otherwise we would have surplus value. They wish also a complete public control of mineral treasures, brought about by leases for the exploitation of these treasures when in public ownership, so that the lessees may receive no more than fair competitive gains; but some conservationists are inclined to look favorably upon at least a limited public

[1] The limits of space and time prevent an elaboration of these theories, and this is the less necessary as I have discussed them elsewhere. See my *Property and Contract*, Book I. Part I, Chap. VI and Part II, Chap. V.

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operation of mines as well as ownership. The distribution of wealth and income is then always included as an aim in thorough conservation discussions.

Conservation Includes Two Orders of Inquiry. The First Belongs to the Natural Sciences

But when we reflect seriously on the subject of conservation, we see that we have to do here with two orders of inquiry: One of them falls within the broad field of the natural sciences; the other is economic in nature and is concerned with property relations.[2] The geologist must instruct us about the formations of mineral treasures, about their amounts so far as actually known and so far as they may be estimated, about exhaustibility, etc. The scientific agriculturist deals with the cultivation of the soil and discusses methods for the maintenance and even increase of agricultural productivity.[3] Men who understand the natural laws governing the growth of trees are absolutely required if we would frame a wise forest policy. It is a question of natural science to determine the relative amounts of forest products which can be expected under low and middle and high forest culture, all involving different periods of time.

The Second Order of Inquiry is Economic

But chief emphasis must be laid on the too much neglected question of property relations. Is the system of leasing public land to private parties favorable in the case of mines? We have to do in this case with the relations of public property to private operation and to private property in the implements of operations. Shall the federal government, the States, cities and other local political units acquire private lands for forestry? Here we are concerned with the extension of public property in land and the corresponding contraction of private property in land. How shall we encourage oyster culture? Shall we make the beds of bays and rivers and bodies of water where oyster farming is feasible private property? If so, we extend private property and contract correspondingly public property. Shall we reserve public property in underwater land and encourage oyster farming and similar production of other sea food by a system of leases which will encourage private investments of capital? Again, we are concerned with property relations. It is in the property relations most suitable for conservation

[2] It is recognized that in certain of its features, conservation has to do with political science and sociology. Here and now we must confine ourselves to economics as that one of the social sciences with which we are chiefly concerned in conservation.

[3] The maintenance of productivity and conservation of elements are two different things, but it would take us too far into scientific agriculture to discuss this distinction here.

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that the chief difficulty arises; and it is on this account that the chief rôle in conservation belongs to the political economists, who must cultivate more diligently than heretofore that part of their field which we may designate as economic jurisprudence. When we decide upon what we want so far as property relations are concerned, engineers and other technical men will be able to carry out the desired policy, and political science, including administration, must be called on for aid. Oyster farming will flourish when we devise and put in force right property relations, and it will furthermore flourish without the establishment of injurious private monopolies.

Irrigation furnishes also an excellent illustration. The government of the United States has encouraged technical investigation. Much effort, and very properly, has been put upon ascertaining the underground flow of water and what is called the duty of water, *i.e.*, the work which a given quantity of water may accomplish, the aim being to avoid waste and accomplish the maximum of effect with the minimum of effort. The questions of when and how to apply the water are involved here; and many other technical questions, some of them purely engineering, some of them primarily agricultural, could be mentioned. These are technical and essential. But litigation has been prominent and in some parts of the country has ruined many a farmer, resulted in mutual hatred of neighbors, and occasionally even bloodshed has been a consequence, while an appreciable proportion of the benefits of improved farming by means of irrigation has been swallowed up in the needless costs, direct and indirect. And the vital question of property relations was neglected in the meantime, one great department, the Department of Agriculture, at one time even taking a discouraging attitude toward economic study. However, this has all been changed under the efforts of the present Secretary of the Department of Agriculture. Now, when property relations are settled, the rest becomes comparatively easy.

The United States Government in the Reclamation Service has not neglected property relations so far as its projects are concerned, for they are based on a rearrangement of property relations in the interest of the public and the parties immediately interested. I refer here to the giving up of old "priorities" so as to promote a better distribution of water.

Did time permit, it would be instructive to follow this discussion of the nature of conservation with a brief historical survey, showing first of all what political economists have done for conservation, and how the ground was prepared for the modern conservation movement; secondly, describing the connection which has existed between the work of scientific foresters, such as Dr. B. E. Fernow, Gifford Pinchot and others with the conservation movement; and thirdly, outlining the services of those men, who under the leadership of Ex-President Roosevelt

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made the program of conservation a vital force in the United States. But as it is, such a historical sketch must be omitted; and so we proceed at once to a discussion of some leading economic principles of conservation.

II. Some Economic Principles of Conservation

Conservation in its economic aspects has such a wide sweep that it is difficult to lay down many principles of universal application, but several can be elaborated which have very far-reaching applications. But before we call attention to certain general principles, we ask on the very threshhold of our inquiries that you should consider this simple question, "What is waste?" The question is simple; the answer is in many cases extremely difficult. The Century Dictionary has over two columns devoted to the word waste as noun, adjective and verb, but does not satisfactorily answer our question. Under the verb, I find this: "to expend without adequate return." That is, at any rate, helpful. You see apples decaying on the ground in my orchard. You ask, "Why this waste?" If I reply, "It will take two dollars worth of time to gather one dollar's worth of apples," are you satisfied that there is no economic waste? But this is precisely the case with a great deal of what the conservationist frequently calls waste. Now I shall not attempt a definition of economic waste, but I hope that the remainder of this paper will throw some light on the question, and will show that often we must choose between destruction of a proportion, larger or smaller, of natural resources and a loss of human well-being; and as the natural world exists for the human world, we know how we must decide when the issue is clear-cut between the two.

Conservation means in a large number of cases what economists usually designate as intensive as opposed to extensive use. Now intensive use means in general high price. This is true in agriculture—excellent fences, careful cultivation of the corners of field and the edges near the fences, if there are fences; also fields entirely free from weeds, which means the hoe as well as the horse cultivator. It means going beyond the point of increasing returns to decreasing returns in most cases. Nearly if not quite everywhere in the United States the intelligent farmer stops when to go further decreases his net money income; this means in a new country very often that the hoe is never used, that the edges of fields are ragged, that fences are not so neat in appearances as esthetic interest would demand.

The Higher the Price of Land, the Better the Farming in the Absolute Sense.[4] Warren in his *Farm Management* comes back again and again

[4] It is the high price of land rather than the high price of the products from the land that is the cause of good farming and intensive farming. It is important to bear this distinction in mind, but we cannot here enter into all the refinements of theory which flow from this distinction. Historically and generally, it is believed that the two go together.

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to high price as a cause of good farming. A soiling system, *i.e.*, bringing the feed to cows, is conservation in the sense that it means more return per acre, but Warren says truly, "The labor of hauling the feed and manure, to say nothing of the cost of growing the crops, would more than pay the pasture bill on most dairy farms. It is evident that land and milk must be very high in price, before a soiling system will pay" (p. 179). And we find the following observation in the chapter on "Maintaining the Fertility of the Land:"

"We have good years and poor years, but crop yields are increasing very rapidly. All that is necessary to have them go up still farther is to pay the farmer more for his produce. By bringing in land that is now little used, and by better methods of farming, that are already known to farmers, it would probably be possible to increase our total production of crops 50 per cent. in three years if the farmer could be assured of prices high enough to warrant the expense involved."[5]

Undoubtedly it is true that without an increase in price, something may be done to increase yield. Sometimes farmers are more intensive than they can afford to be, but generally not so much so. The following from Warren is a correct generalization. "The farmer's problem is to intensify his business up to the point of greatest profit for his conditions. Since conditions are gradually changing in favor of more intensive methods; and since there is a tendency for the average person[6] to lag behind, it follows that a little more intensive methods than the average of the community will usually be best" (pp. 179–180). But observe methods are changing to favor intensive farming because land prices are higher.

Another thing to be noticed is that improvement in knowledge and art increases the profitable intensivity of farming. We learn how to increase yields at a given price.

In what has been said we have a partial explanation of the intensive dairy farming which may be seen in parts of Europe, *e.g.*, near Frankfort-on-the-Main. But we have only a part of the explanation. Low wages are another part of the explanation; for, other things being equal, the higher wages are, the less intensive is that farming which is profitable; the lower wages are in a given state of the arts, the better the farming which is profitable.[7] High wages encourage the use of machinery, and this is a partial offset, but only a partial offset to high wages. If wages could be cut one-third and the labor supply increased correspondingly, we should witness an improvement in farming which would be in the direction of conservation of natural resources and would please the

[5] Warren: *Farm Management*, p. 183–184.

[6] It is not a question of averages. Many lag behind and for all these better farming would be profitable.

[7] High prices and low wages result in increased intensivity by drawing rent up. It is high rent with high land values that makes intensive farming profitable.

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esthetic eye; but observe it is a conservation of natural resources which is at the expense of human resources.[8]

The Conservation of Human Resources Limits the Conservation of Natural Resources

Now this can be illustrated endlessly, in mining, as well as in agriculture. If we save all the ore taken from the mines in the Mesaba region in Minnesota, the price of iron ore must increase very materially or wages must fall, or both effects must be produced and either one means lessening real incomes of the community at large. As in farming, somewhat more economical methods could doubtless be employed at present prices for ore and present wages; but at best great waste of natural resources must continue, *i.e.*, a great deal of the ore must remain unutilized.

Conservation of natural resources everywhere finds its sharp limitations in human resources, *i.e.*, in human well-being.

Once again, conservation means a sacrifice of the present generation to future generations, whenever it is carried far, this conflict beginning far before the point is reached which conservationists are inclined to advocate.

This is implied in what we have already said. High prices and low wages, or even high prices with present or higher wages, mean a sacrifice of the interests of the present consumer. How shall we balance the interests of the present and the future? If we put them on absolutely the same plane, we could with propriety forego all use of the exhaustible natural resources like petroleum and natural gas; it would be at any given moment of time indifferent whether or not we used them, and they might remain forever unused—a *reductio ad absurdum.*

Perhaps the best article on the economic theories involved in conservation is that written by Professor L. C. Gray, which appeared in the *Quarterly Journal of Economics* for May, 1913, under the title, Economic Possibilities of Conservation. Professor Gray finds "the real heart of the conservation problem" in "the conflict between the present and future" (p. 499). He says that "the primary problem of conservation expressed in economic language, is the determination of the proper rate of discount on the future with respect to the utilization of our natural resources." He then continues as follows:

"Some discount of the future, there must be. If society reduced the discount on the future to zero, the period of utilization would be increased to infinity; and there-

[8] Economic theorists will want to elaborate further and qualify the statement. High prices and low wages would in some cases result in larger scale and less intensive culture; but what is said here holds as a general statement. Rents would usually increase. Movements of population must be considered if we are to make our theory complete; but to do this would mean an immense enlargement of the present paper.

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fore, the amount of present use would become infinitesimal. Conservation as a single principle of action involves the equal importance of future wants and present wants. It requires that the want of the infinitely distant future shall be as important as the want of the immediate present. Conservation as a single principle of action is reduced to an absurdity" (p. 515).

This is precisely the conclusion we have reached. Any positive, statistical and mathematical solution is an impossibility, because the solution, as Professor Gray also points out, depends in the final analysis upon questions of individual social philosophy. What is the purpose of existence? the source and nature of ethical obligation? our duties to posterity? all questions of the gravest import and beyond the range of economic science. But taking as a basis the ethical notions and sentiments of normal men, or perhaps those somewhat above the average man, we can find guiding principles and helpful suggestions in economic theory which we must apply as best we can in concrete cases in their infinite complexity.

Now, as one of the first steps in conservation policies, we must classify natural resources, because they differ markedly with respect to the intensity of the conflict between present and future interests. Some natural resources may be maintained forever or at least indefinitely with use; others may possibly be increased indefinitely while being used; others show increasing scarcity and exhaustibility; and where supply is so sharply limited in proportion to demand that we begin to feel the effects of coming exhaustion and where no renewal is possible, we have the sharpest conflict between present and future. Let us for our present purposes adopt the classification of Professor Gray, which as made by an economist is especially adapted to the purposes of economic discussion.

"Natural resources may be classified as follows:

"I. Resources which exist in such abundance that there is no apparent necessity for economy, either in present or future. For instance, water in some localities.

"II. Resources which will probably become scarce in the remote future, although so abundant as to have no market value in the present. For instance, building stone and sand in some localities.

"III. Resources which have a present scarcity:

1. Not exhaustible through normal use: Water powers.

2. Necessarily exhausted through use, and nonrestorable after exhaustion: Mineral deposits.

3. Necessarily exhausted through use, but restorable: Forests, fish.

4. Exhaustible in a given locality but restorable through the employment of other resources of a different kind or of similar resources in different locations: Agricultural land"[9] (pp. 499–500).

[9] In a footnote Professor Gray says: "In terminology the above classification resembles one proposed some years ago by Professor B. E. Fernow: See his Economics of Forestry (p. 10). In detail, however, the classification differs widely" (p. 500).

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Interest represents the difference between present and future to the individual. It has been said in the economy of the State, as in publicly owned forests, interest does not enter, that it is a concept in distribution of income among private persons; but, however this may be, we cannot avoid the recognition of differences between present and future. Can it be asked that we should do more than discount the future at the lowest possible rate of interest paid by a prosperous state with well-managed finances, say 2 per cent.? I throw this out as a suggestion. It is only a little bit of a sidelight on the complicated problem.

It is essential at this point that we consider the case of copper and other natural resources which are probably fundamental conditions of permanent national existence. Even if there is some doubt about this, so long as it is at all probable that any national resource is essential to the continued and permanent existence of the nation, this resource occupies a peculiar condition. It is a first principle of political science that the State has perpetual life. States have perished in the past, but political and economic science cannot take into account the possibility that our own national life will ever cease to exist. All wise plans must be based upon the hypothesis of continued national existence. Now in a case of this kind the future value of the natural resources rises to infinity, and however much we discount the future, we must still practise conservation. We must attempt so to utilize every natural resource of this kind that it may last as long as possible, using it now and using it hereafter for as long a time as possible. I think there in reality the interest rate cuts no figure. When we consider resources of this sort, we find that in their case the definition given by President Van Hise at the close of his book is applicable: Conservation means "the greatest good to the greatest number—and that for the longest time."[10]

There has been some discussion as to whether in agriculture we want the largest product per man or per acre. In general, it is doubtless true that the chief determinant should be the quantity produced per man. But the principle just laid down leads to certain qualifications. We have to consider not only the quantity produced per man now, but the welfare of future generations, and especially must we consider the condition of the country at war. The conservation of natural resources has to be regarded from a national point of view. No nation can consider it otherwise at the present time. From the German point of view we can see that it was important that Germany should sacrifice, as she did, something from the largest possible production per man to increase the production per acre. Conditions in Ireland also show that in the interest of present and future generations we cannot decide solely with reference to the largest production per man.

[10] C. R. Van Hise: *The Conservation of Natural Resources in the United States*, p. 329.

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Adequate conservation in general means a course of conduct in economic affairs which is dictated by the common interest, but which will not be followed by the private persons under a system of *laissez faire*, or nonintervention.

The complaint of the conservationists is precisely this: that the individual does not, as a matter of fact, conserve natural resources. What is going to induce the private person to follow the desired course of economic conduct?

We consider among various possibilities this: the individual through ignorance fails to follow a socially desirable course of economic conduct, when it would be in his own interest. We have already seen that often, perhaps in most cases, the individual could be more careful, more painstaking, more intensive in the sense of conservation—for not all intensive uses mean conservation. We have, as a remedy against ignorance, education, and education can discover many new ways of conserving, to a greater degree than at present, all natural resources.

Even with respect to the present, education, if properly directed, will result in a diminished waste of economic goods. Psychical development fails to keep pace with economic changes. A highly educated woman, a missionary who had long worked in Turkey-in-Asia, was asked by a ladies' society in Buffalo what impressed her most on her return to her native land, and she replied, "Your garbage pails." She saw in them sinful waste—good food thrown away in a single city, which would nourish thousands and perhaps even ten thousands of half-famished people in Armenia. Several causes of American waste in this particular can be mentioned. One is that when food was superabundant, that was not waste which is now wicked waste. It is not wasteful to feed potatoes to pigs when the supply is greater than the needs of all human beings who can be reached. We have in general left behind us the days of crude plenty, but have not changed our psychical make-up nor our habits to correspond with new economic conditions. Here the need is intellectual and moral education—a better vision and more altruism. We need a keener social consciousness and a new state-sense, if we are ever to solve the problems of conservation.

The individual is too frequently thoughtless and indifferent with respect to the future. Education of the intellect helps to a slight extent, because as men become more intellectual, the future means more to them. Moral education helps still more, as it means precisely a strengthening of the other-regarding feelings as opposed to self-regarding impulses. It means more self-control and ability to sacrifice within rational limits the present for the future. Better government means that the future counts for more, as it is more certain. Better health and long life further decrease the premium which the future must pay for present sacrifices. Without adding further considerations we may generalize

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as follows: Every step forward in civilization means increased regard for the interests of the future.

But, on the other hand, we must acknowledge that enlightened self-interest has a greater rôle to play in conservation than is generally understood. When attention 15 years ago was called to the enormous waste of natural resources, the private owners began to think about the possibilities of profitable conservation, and they are still making investigations and improving methods. Great mining companies particularly may be mentioned as illustration, and naturally the owners of the fee in mining properties have shown a special concern. Real progress has thus been made; but the rate of interest sets a limit, as my colleague, Professor Leith, has shown. Some ore companies have been obliged to recede in part (in part only, observe) from earlier conservation practices, because they found no prospect of getting interest on the investment involved in carrying saving so far as they were doing.

Large concerns are apt to be operated by far-seeing men, who acquire at times an interest of affection, so to speak, in these concerns, and they will be inclined to go as far as is practicable in conservation; as the country grows older, the interest rate is likely to fall and other causes will favor an increasing degree of conservation, due to private initiative.

But there is a sharp limit to the economic sacrifice that we may reasonably and ethically ask the private person to make for even the present welfare, and the limit is still sharper when we come to consider the interests of future generations. When it is possible and as a general principle, social burdens should be socially diffused and socially borne.

Let us consider some application of the principle we have just formulated. We know now that it is one of the prime functions of the State (State used in the generic sense and including federal government as well as the separate commonwealth) to raise the ethical level of competition by legislation and administration. Men may work 12 hours a day under the system of *laissez faire* and their employers may compete with each other. If, by the State, the number of hours is reduced to eight, the competition of employers may continue unchanged, save that it takes place on a higher ethical plane. Endless illustrations could be given, taken from Sunday work, child labor, sanitary conditions, etc. Now we may similarly establish a level of competition based on the principles of conservation. Practices which are not in accord with that level of conservation, which we wish to establish, may be prohibited. When we decide how carefully the coal should be mined, and how much waste of natural resources in the exploitation of iron ore we are willing to tolerate as a maximum, we may prescribe methods giving the desired results, and we must bear the pain or burden in higher prices. Competition

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would not allow one individual to increase his costs greatly above those of his fellows; and so common action is a necessity.

The mention of competition suggests that unregulated and uncontrolled it is very generally speaking destructive alike of human and natural resources, and that it is a force which like the flow of rivers needs to be bridled and guided to produce socially beneficent results. Destructive competition can be prevented only by social efforts. Again, I must mention the work of President Van Hise in pointing out in his book, *Concentration and Control, A Solution of Trust Problems in the United States* (1912, revised edition 1914), the evils resulting from those "trust-busting" campaigns of politicians who advocate competition at all times and places and let loose upon us forces of destruction.

But as the principle of regularities in large numbers forms the basis of the vast business of insurance, it also furnishes a considerable scope for the activity of the State in the conservation of human natural resources. It is well known that individuals, especially among the wage-earners, are too willing to take risks to life and limb. Let us suppose the chance of becoming a helpless cripple in the course of a year is as 1 to 10,000; this seems so small that the individual is generally ready to overlook it, and neglects measures to remove the danger. But to society there is no chance, only certainty of this human loss, which in its ramifications is far greater than is at first apparent. It means 100 wrecked existences per million and 10,000 per annum in a population of one hundred millions. We have in this principle of regularities in large numbers a wide scope for society to promote conservation alike in its own interest and in the interest of the individual. The "Safety First" movement is one of many evidences of an awakening realization of possibilities in this direction.

As another illustration of the possibilities of altogether wholesome conservation which comes under this general head I mention the enormous destruction caused by fires in the United States. Public authority must compel those precautions which are alike in the interest of the individual and of society.

But we may look at this question from still another angle. *Protectionism has often an educational value.* When we prohibit certain practices which exhaust natural resources rapidly, it will often be found that through the pressure of legal force we shall be stirred out of a certain inertia of lethargy, and discover better methods which involve little or no increase in expense.

But we find ourselves in a position when we shall frequently have to decide between private property and public property. The benefits of private property come in large part from the free initiative it allows the private property owner, and this is based on the hypothesis that in

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following his own initiative and knowing his own interest he is following a line of conduct which is in the interest of society. This is illustrated by the condition of the farmer, where we have a fairly satisfactory system of land tenure. The intelligent farmer in pursuing his own interest, in raising the best crops and animals at a minimum cost, is doing exactly what it is in the interest of society he should do. The case of our railways is the opposite, for we regulate these to such an extent as to remove from private property a large part of its satisfactions and its benefits. So if we are obliged to regulate very far private property in the interest of conservation, we have a strong ground for public property; as illustrated in the case of forests, and in this case public ownership is the world over gradually gaining on private ownership. Consideration of conservation then leads us to the following:

"*The principle of guidance in changes from private to public property and from public to private property.*—Private property yields the best results when the social benefits of private property accrue:

(*a*) Largely spontaneously.

(*b*) When occasionally they are easily secured by slight applications of force.

(*c*) When the social benefits of private property are secured as the results of single public acts occurring at considerable intervals.

(*d*) Private property may yield excellent results, when in more or less frequent cases a continuous and considerable application of force may be needed to bring its management up to a socially established ethical level.

"But in proportion as the social benefits desired are secured by increasingly intensive and increasingly frequent applications of public power, the advantages of private property become smaller as contrasted with the advantages of public property."[11]

We have furthermore a ground for passing over to public property when for the sake of the public welfare now or in the future a kind of use is exacted, involving a greater sacrifice of individual and corporate private interests than is warranted by the doctrine of the police power, as accepted at a particular place and time.

I cannot go into all these points, but I am confident that the more you think about this statement of a general principle, the more far-reaching you will see that it is in its application to conservation. I want to say, however, just a word about the last clause, which relates to the police power. We hold all our private property subject to that; in other words, certain general sacrifices may be exacted of all in the general interest, relating to health, decency, etc., and this burden increases as time goes

[11] See Ely's *Property and Contract*, vol. i, p. 352, where an elucidation of this principle is given.

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on; but when we must exact more than is in harmony with this limitation, known as the police power, we have a ground for public ownership.

No one is warranted in jumping to the conclusion that this principle means a suppression of private ownership of land and capital and of private initiative. On the one hand, our classification of natural resources shows that, so far as we can now see, they are mostly adapted to private ownership; we have also observed improved practices of private owners, rendering less urgent and less far-reaching the demands of public ownership; on the other hand, the growth of the police power in itself makes the field of public ownership narrower than it would be otherwise. Yet the general principle holds and no one can now foresee where the line will be drawn in the future. In the case of forests, the civilized world now recognizes a large amount of public ownership as necessary; so also with regard to the shores of harbors; so also with regard to mineral treasures, where there is general opposition to further alienation of the fee where it is now in public ownership.

Conservation then necessarily means more public ownership, more public business; this means a demand for better government; and this means giving men a real career in the public service.

It would require a small book—perhaps a large one—to elaborate the principles I have hastily sketched, and to fill in the gaps and complete the argument would certainly require a very big book. But I trust I have outlined the field for the economist in conservation.

Commissions Alone Equal to the Tasks of Conservation

Now I will close with one very practical suggestion. It is only through a conservation commission or various conservation commissions—for several may be required—that we can put in force conservation. Legislation can never solve these complex problems, but can simply lay down the general principles, expressive of the will of the legislature; and in concrete cases the commission must make the application of principles, in other words, ascertain the will of the legislature. The commission must ascertain what is excessive present use and what is waste, must in concrete cases weigh over against each other present and future, must decide upon the burden to be imposed on private property under the police power, and upon compensation which is feasible and required; it must set limits to the sacrifices which may be legally and ethically exacted of the individual and the private corporation.[12] From

[12] On the sense of social responsibility in its relation to conservation and the sacrifices to be enforced by law on the individual and the private corporation, see Van Hise: *The Conservation of Natural Resources in the United States*, pp. 362–3; 374–5; 377. I think President Van Hise exacts too much of the private individual.

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time to time it will have to recommend the establishment of new principles by legislation; and the courts, as having the last word in social progress, will review certain decisions.

But the great burden must rest with commissions. And do not commissions give us the democratic solution of the complex economic problems of our day? The legislature lays down the principles, and if the commissions furnish men with careers, with honor, the public may command as well-trained capacity and as high talent as the greatest private corporations and for far less cost. Recognition, the development of democratic sources of honor, open careers to capacity and talent; all these will draw to the service of the people men who will be equal to the tasks of government, and who in their own persons will illustrate the nobility of social service and make men proud to say: "I am a civil servant."

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The Use of Low-grade Phosphates

BY JAMES A. BARR, B. S., E.. M.,* MT. PLEASANT, TENN.

(New York Meeting, February, 1916)

WHEN phosphate mining operations first commenced in Tennessee the loss of both high- and low-grade material was large, because of the crude hand methods employed. Practically all rock smaller than 2 in. was thrown back into the pits and covered with the overburden of subsequent openings.

With the advent of mechanical washers and other mining machinery, together with the abolishing of the contract system, the waste has been steadily diminished, until at present only the finest sand is being rejected; and even this is not lost, for in the majority of cases the water, slime, and sand from the washers are run into large settling ponds where practically all of the phosphate-bearing material is caught and stored for future use, or until it is profitable to ship lower grades.

The latest equipment used to recover the fine and lower-grade phosphate sand includes Dorr thickeners, settling tanks, Dorr classifiers and settling ponds served by locomotive cranes with clam-shell buckets, making it possible to save material averaging as fine as 200 mesh. To attempt saving finer material would necessitate a different type of drier, which might be patterned after the multiple-deck roasters.

The present tailing runs from 15 to 25 per cent. of available phosphate, which, however, as mentioned before, does not represent a total loss to possible future operations. The demand of the consumer is the greatest factor in regulating the recovery at present.

In the mining of blue rock, which does not require washing treatment, the main waste material is the capping of low-grade kidney formation, generally used for backfilling of the rooms.

By reason of the introduction of improved methods in the early stages and the re-mining of territory skimmed over at the start, at the present time the net result of all operations is a comparatively small loss.

One of the largest temporary losses in the industry occurs in the Florida pebble district where the recovery of phosphate runs about 25 per cent. The loss is classed as temporary because most of the tailing has been collected in ponds and piles, where it may be reclaimed in the future should developments or improved processes warrant.

In cleaning the Florida pebble phosphate, aside from the elutriation

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of the clayey binder and strata that sometimes contaminate the rock, the main operation is that of sizing, by which the large phosphate grains are separated from the smaller silica sand. This results in the loss of all the fine phosphate along with the silica. Owing to the similarity of the two in specific gravity, no mechanical process has been developed as yet to effect a further saving.

As the problem now stands, the development of some concentration process is required to effect a further recovery of the low-grade phosphate in Tennessee and Florida, where the consumers demand a high-grade product.

Taking 68 per cent. tricalcium phosphate as the present minimum average grade for Florida pebble and Tennessee blue rock, and 72 per cent. for the Tennessee brown rock, it will be seen that the development of such a concentration process is a much needed step in the conservation of the phosphate resources, for by its introduction the commercial limits for use in complete fertilizers could be reduced to 60 per cent. and possibly 50 per cent. tricalcium phosphate.

In the event of low grades of phosphate becoming commercial products, 20,000,000 tons would become available within a 50-mile radius of Mt. Pleasant, Tenn., alone, and perhaps 100,000,000 tons of rock, especially if phosphatic limestone is taken into account. These figures are not claimed to be precise, but are given to indicate that the available phosphatic resources have, so to speak, merely been scratched.

It would seem that the most feasible means for making low-grade phosphate commercially available lies in the development of a smelting or electrometallurgical process whereby the phosphorus content of the rock would be rendered available (soluble) directly and without the introduction of a diluent, as in the manufacture of acid phosphate by the sulphuric acid method. By such a direct conversion, a 60 per cent. rock would serve the same purpose as a 72 to 75 per cent. rock with the sulphuric acid method.

Along these same lines a combination process may be developed whereby low-grade phosphate and an insoluble potash feldspar or similar potash mineral may, by heat or other treatment, be rendered available for use as a commercial fertilizer. The European war has brought out the importance of such a discovery.

Again, there might be developed another combination process whereby ammonia phosphate could be made directly, as cynamid is now manufactured with the aid of an electric arc.

The greatest possibility of using low-grade phosphate lies in the direct application of finely ground rock to the soil in the raw state, allowing the acids of nature to convert it into soluble forms. The limiting factor in this case is the freight charge on the inert and nonphosphatic portion.

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The difficulty with the ground-phosphate trade has been that the fineness of grinding was not such that a sufficient percentage of the commercial product became available within any reasonable time. When the grinding industry was new, the fineness was generally 90 per cent. through 80 mesh, while now it is 90 per cent. through 100 mesh, or even finer. The impalpable powder gives effective and immediate results; therefore the future extension of this particular phase of the phosphate industry will be aided greatly by finer grinding.

Another available source of phosphorus is basic slag from steel manufacture. The United States Steel Corporation has erected a plant in Alabama to prepare such slag for fertilizing purposes and has placed the product on the market. This form of fertilizer has been successfully used in Germany.

An average grain crop requires and takes about 15 lb. of phosphorus per acre from the soil in which it is grown. This phosphorus must eventually be replenished, for with a meager amount of phosphorus in the soil the crops are poor, and with none the harvest is nil. The majority of farmers fail to replenish the soil with the elements taken out by the crops, and at no very distant time they will be drawing heavily on our phosphate resources. In view of this, the use of the lower grades is certain to become common.

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Pennsylvania Fire Clay

BY L. C. MORGANROTH, PITTSBURGH, PA.

(New York Meeting, February, 1916)

Carboniferous Clays

From a geological standpoint, but scant attention has been paid to fire-clay beds. Only within the last few years have they been the subject of individual investigation, prior to this time having been considered only in their relation to coal seams. While the relative importance of coal and fire clay is still immeasurably in favor of coal, yet an industry as large as that of fire clay is entitled to a measure of attention.

A discussion of the Carboniferous clays is practically a discussion of the clay beds in the Appalachian basin, since more than 90 per cent. of these clays mined are from this region. The clays are confined to the northern end of the basin, or the States of Pennsylvania, Ohio, Kentucky, Maryland and West Virginia, ranking in importance about in the order named.

The clay deposits of the Carboniferous period have been divided into two classes; those found in the Pottsville conglomerate series of rocks, and all those that are found in the formations above, for the reason that the clay beds above the Pottsville series all resemble each other in properties, composition and appearance, and all are distinctly different from the bed found in the Pottsville formation. The bed of clay found in the latter formation is unique in that flint clay is practically restricted to it. (A variety of flint clay is occasionally found in these upper deposits but this is in such small amounts as to be negligible.)

Carboniferous clays occur between stratified rocks; sandstones, limestones, etc., in the same manner as coal veins. They are often accompanied by a coal vein although where the clay is of a workable thickness, the coal is generally too thin to work. The clay beds are sometimes known by the name of the coal seam at which horizon they occur. Clay beds may be from a few inches up to 20 ft. in thickness. They may consist of all soft clay, all flint or both soft and flint.

The Carboniferous clay beds of Pennsylvania are the most important of any of the clay beds found in the country, and are among the oldest ones worked. We know more about this clay geologically for the reason

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that Pennsylvania coal veins are better known and, as previously stated, the clay beds accompany coal veins.

Pottsville Conglomerate Series

In the State of Pennsylvania there are 16,000 square miles containing the coal-measure formation. It is within this area that the Conglomerate clay is found.

Only a comparatively small area of the Pottsville Conglomerate series lies above water level. While clay of good quality is comparatively scarce, it is still sufficiently plentiful to make it unprofitable to mine the clay below water level because its great irregularity would necessitate a large and unwarranted development expense.

The accessibility of a clay field is another point. While large bodies of clay of the proper quality will warrant a considerable expenditure in railroads to reach it, there are limits; the average required to supply any ordinary sized brick yard is not large, and unless other freight could be handled on the same road, it is doubtful if it would pay when the length of track to be built is over 10 or 15 miles. With the above points in mind, it can be understood why the various clay mines are grouped within comparatively narrow bounds.

It is not to be supposed that the entire area within the Pottsville Conglomerate is underlaid with clay. No doubt clay beds suffered as great a loss from erosion by streams and glaciers as did the coal veins, so that today we have but a small percentage of the clay originally deposited. Again, the clay beds never covered as great an area as the coal veins. Their sedimentary origin would preclude this idea. While we look for clay within the boundaries of a certain formation, it is not to be supposed that it will be found anywhere within this boundary; the major portion of the area probably never contained any.

We know of no clay being mined in the Pottsville series of the anthracite field. It would not pay to work it by itself, if found, for it would nearly always be below water level. In the counties in the north and northwestern part of the State where the Pottsville Conglomerate outcrops, there are localities where the clay shows, but as a rule, the clay is not good. Other counties, it can be seen, will not pay even to investigate on account of lack of railroad facilities.

The bulk of the Conglomerate fire-clay mines are in Clearfield, Clinton, and Cambria Counties, with a fewer number in Somerset, Bedford, Clarion, and Armstrong. The most valuable bodies of this clay are those found in central Pennsylvania, viz.: Clearfield, Cambria, and Clinton Counties. These counties rank in the order given, in reference to quality and quantity of the clays produced. In a general way, the clays of central Pennsylvania are about the same over the entire area;

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they look alike, analyze about the same, and are similar in formation of the deposit, while the brick made from them are about the same in refractoriness; yet in this area there are clay bodies that possess distinctive features and must be treated differently in their use. Chemically, the difference may be slight, if any, yet must be taken into account by the manufacturer. Clearfield County particularly possesses several bodies of this Conglomerate bed which show marked differences.

It may be that some of the Conglomerate beds now worked occur at different geological horizons, but it is impossible to determine at present if such is the case. All the Pottsville Conglomerate beds now worked in Pennsylvania are considered as occupying the same geological position, and only when the fields have been more thoroughly prospected will this fact be determined positively.

The exact position of the bed is supposed to be directly beneath the Upper Mercer Coal.

Probably the most important clay field in Pennsylvania, if not in the United States, is that at Woodland, Clearfield County, Pa. These mines are among the oldest, if not the oldest, in the State. The brick made from them are probably more widely known than any other brand. The Woodland clay is noted for its purity and for its regularity in quality. Other mines may have as good clay at times, but no mine can show for the same number of years the same regularity in quality. It has also been one of the most regular in regard to quantity; while it has been worked for a number of years, the operators claim there is more clay in sight today than there was 10 years ago.

One of the principal reasons for the superiority of the Woodland clay is the excellent quality of its soft clay. Other Conglomerate beds may have good flint, but it is the exception for them to have any amount of good soft clay. The Woodland vein averages 2 ft. of both flint and soft clay. The clay outcrops for but a short distance, probably not more than 2 miles in length. A stream running east and west cuts across the field so that there are two crop lines. Its exposure at Woodland is due to a north and south anticline which has lifted the measures above water level. The dip of the anticlinal east and west is rapid, so that in a short distance the clay horizon is 100 ft. or more below water level.

Two other important clay fields are found in proximity to the Woodland bed; these fields are known locally as the Morgan Run field and the Andrews Creek field. The former lies about 12 miles south of Woodland and the latter about 10 miles west of it. A circle of 2 miles radius will more than embrace either field. Geologically these veins occur at the same horizon as the Woodland clay but the clays differ from that of the Woodland bed. The clay of the Morgan Run field is rougher in appearance and looks more like burley than flint. The bed is thicker, but there is very little of soft clay, so little that the vein is often mined "run-of-

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mine" and no effort made to keep the two varieties separate, as in the Woodland bed. The bed when used run-of-mine seems to have a right amount of bonding property. The appearance of the bed, and the fact that it is so used, indicates that the clay is not as hard as that of Woodland flint and because of this, the small amount of soft material found in the bed is sufficient to make the bond.

Anderson Creek Field.—The clay of the Anderson Creek field is distinctive inasmuch as the bed has all the appearance of a soft clay, or rather a "hard soft." It does not resemble the burley clay of the Morgan Run field nor the Woodland soft or hard, but appears to be in a class by itself. While it has not the flint-like nature of the hard and while closely resembling soft clay, yet it is as deficient in bonding properties as the best flint. Chemically, it is as pure as flint and in tests almost as refractory. A portion of this same field, locally known as the Widemire field, is a distinct variety of clay. A considerable amount of nodule clay is found in this field.

There are several other clay fields in Clearfield County which are known by local names, but they possess no distinctive characteristics. They resemble the clays of the other fields that have been discussed; in fact, the fields described resemble each other very closely; in any of them can be found clay similar to the others. The differences herein mentioned are noticeable only by close scrutiny. In a comparison with clays of other sections, the clays of Clearfield County would be taken as a whole.

Cambria County Field.—The clay mines in this county are restricted to a few localities, Dean and Blandburg being the principal mining centers. The deposit at Dean is almost all flint. There is, on an average, about 6 in. of "soft" below it, but it is often impure and is not used. The vein at Dean appears to be the flintiest and hardest of all the clays of Pennsylvania, if flint clay is a hardened form of fire clay, as previously argued; at this mine it has reached its highest development, as practically all the vein is flint clay.

The vein at Blandburg so closely resembles Woodland that little more need be said about it. There is a locality in this section where the vein consists of from 7 to 10 ft. of nearly all "soft." This soft is practically the same as the Woodland soft.

Clinton County Clays.—There are a number of separate localities where clay is mined in this county, but the character of the deposit is practically the same and one description will apply to all. The bed consists principally of soft clay; the flint clay averages from 12 to 18 in. thick. The soft clay is generally 4 to 6 ft. in thickness, and in some places even thicker, but it is not all good. It rapidly increases in impurities as the bottom of the bed is reached. The flint clay is probably as good as any, but it is the soft clay that makes these clays rank below

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the best. Generally, about 1 ft. of "soft" immediately below the flint is very good, but if only the flint and this 1 ft. of soft clay were mined, the clay would be too expensive to use. As it is, a portion of the inferior "soft" is also mined to give a vein of workable thickness. Of course, there are exceptions to this statement, and mines are found where excellent soft clay is mined, but as a general rule the soft is impure. The impurity is iron.

Clarion Field.—This field takes its name from the county in which it is found. The field produces a very pure flint clay, but the clay of this grade occurs irregularly in pockets. A considerable portion of it is used in glass pot construction, some of it being calcined at the mines for this purpose. The clay is lighter in color than other Pennsylvania flint clays. Another peculiarity is the large amount of iron balls found in the vein; they range in size from that of a walnut to a football, and many are perfectly round. The amount of clay shipped from this field is dwindling, due to exhaustion.

Climax Clay.—In the vicinity of Climax and St. Clarles on Red Bank Creek, Armstrong County, flint clay is mined. While this field is close to the Clarion fields, the vein more resembles the Savage Mountain clays. The vein is from 6 to 12 ft. thick, with from 2 to 2½ ft. of flint. It is more pockety than Clearfield County clays. The flint occurs more in the form of lenses in the soft clay, in a similar way to the upper-bed flints. While the soft clay is generally below, it may be on top, or both on top and bottom of the flint. Often the flint is missing entirely, and the bed is all soft.

Savage Mountain Clay.—This is the name applied to the bed of clay occurring in the Pottsville series in the southwestern part of the State. The clay field is a canoe-shaped one with its northern extremity near Williams Station, Bedford County and extending in a southwest direction into Maryland and West Virginia. The bed is distinctly of basin form dipping toward the center axis east and west as much as 25° in places. At some of the mines in this district, long tunnels are driven to meet the bed. This is possible as the outcrops occur high up on a mountain. By coming down the side of the mountain several hundred feet and driving a tunnel, the clay is cut at a lower level. It is exceptional in clay mining to go to this expense, as the irregularity of clay deposits does not warrant it. The axis of the basin also dips toward the south. Geologically the clay is supposed to occupy the same horizon (Upper Mercer Coal) as the flint clay beds already mentioned, but the clay differs in appearance from the clay mined in central Pennsylvania. Physically it has a banded or foliated appearance which makes it resemble the so-called upper beds of flint clay that are occasionally found in central Pennsylvania. Chemically it runs higher in silica than central Pennsylvania flints, and is less refractory.

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Upper Measures

Pennsylvania.—The plastic clays that accompany the coal veins above the No. 12 series are probably nearly all worked, more or less—if not for firebrick, for sewer pipe, building brick, stoneware, etc.

One of the most used of these clays is that occurring beneath the Clarion coal. It is mined extensively on the Allegheny River near the town of Kittanning, the clay in this locality being known as the Allegheny or Kittanning clay, and in Clearfield County near the towns of Bigler and Wallaceton, where it is known by the name Bigler and Wallaceton clay. The bed is from 6 to 12 ft. thick—all soft clay. Here and there a small lens of flint clay is found, but so infrequent and the lens so small as to be of no importance. The upper and middle portions of the bed are most pure; the lower often contains an excess of silica and considerable iron. In the manufacture of firebrick this clay is used mostly to serve as a bond to hold the more refractory flint clay together. The Allegheny plastic clay is used extensively in the manufacture of building brick—it makes a particularly white brick.

Beaver County.—The clay bed accompanying the lower Kittanning coal is by far the most important. It is the bed mined along the Ohio River in Pennsylvania, Ohio and West Virginia. The bed may be from 6 to 12 ft. in thickness; the upper portion ranging from 2 to 6 ft. is the purest and is that used in the manufacture of firebrick. Firebrick made exclusively of this fire clay would rank as a No. 2 or No. 3 firebrick. The lower portion of the bed when mined is used in making sewer pipe, stoneware, building and paving brick.

In this region, as well as in Ohio and West Virginia, clays occur at the horizons of the Brookville, Clarion, Middle Kittanning, Upper and Lower Freeport coals, but they are not worked on account of the prominence of the Lower Kittanning vein.

Gorman Clay.—Flint clay occurs occasionally at the horizon of the Gorman coal—between the Upper and Middle Kittanning coals—in Clearfield County. This is generally the bed referred to, in that locality, when speaking of the upper bed flint. The bed is entirely too irregular and uncertain as to quality and quantity to make it a profitable commercial undertaking. It may show a workable thickness on the outcrop, but rapidly thins down to nothing or changes into a soft clay.

Upper Freeport Clay.—Often known as the Bolivar fire clay from the town of Bolivar where it has been worked for a number of years. It is also mined at Salina, Pa. A characteristic of the bed is the large amount of iron it contains. Often the iron is in the form of concretions or balls, in which form they can be readily separated from the clay because of their color and weight. At other times the iron shows up as a stain on the clay, or is so finely divided as to make large portions of the bed

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worthless. Some flint clay is found but it varies greatly as to thickness. The flint occurs in lenses and may be at any part of the bed, top, bottom or middle, or be absent entirely. As is usual with the upper bed flints, its analysis shows an excess of silica.

Pittsburgh Coal—Fire Clay.—At the top of the workable section of the Pittsburgh coal occurs a bed of clay 6 to 12 in. in thickness, that is used in places in making a low-grade refractory material. The bed of clay is too thin to work by itself. It is only mined in conjunction with the coal. While thin, the bed extends over a wide area. There are several manufacturing plants along the Monongahela River making use of it. The clay is known locally as horseback.

Discussion

David B. Reger, Morgantown, W. Va.—I notice in the third paragraph of the first page of the paper, that in mentioning the clays of the Pottsville series, the author says: "The bed of clay found in the latter formation is unique in that flint clay is practically restricted to it. (A variety of flint clay is occasionally found in these upper deposits but this is in such small amounts as to be negligible)." If the author wishes to confine himself to the fire clays of Pennsylvania, as the title seems to indicate, I have nothing to say, but if he wishes to include a discussion on fire clays of the adjoining States which the subsequent paragraphs of his paper seem to indicate, then I wish to call attention to the fact that there are large and extensive deposits of fire clay in the Allegheny series, immediately overlying the Pottsville in the northern counties of West Virginia.

These fire clays have been studied carefully, and described in several reports of the West Virginia Geological Survey, first in the report on Ohio, Brooke, and Hancock Counties, in the northern part of the State; described later in the report on Monongalia, Marion, and Taylor Counties, and in the report on Preston County, which followed.

In these latter counties (Monongalia, Marion, Taylor, and Preston) there are some beds of fine fire clay, varying in thickness from 6 to 10 or 12 ft., with a content of only 2 or 3 per cent. of fluxing ingredients, and these are of the typical Savage Mountain appearance and also have the typical Savage Mountain fracture.

They have been prospected, but not mined a great deal. Several of them have been tested in blast furnaces and furnaces for other purposes, in glass-manufacturing plants, also for firebrick, and have been found to be very good.

They are simply awaiting development, and I merely want to call attention to the fact that there are plenty of them in the Allegheny series still above the Pottsville in West Virginia. There are two or

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three valuable seams which I have seen in person and have had analyzed in our laboratory.

L. C. MORGANROTH, Pittsburgh, Pa. (communication to the Secretary*).—It was not intended to give the impression that the flint clays of the United States are restricted to the Pottsville Conglomerate series. The flint deposits of Missouri are an exception; in Tuscarawas County, Ohio, at the horizon of the Lower Kittanning Coal, occurs a valuable bed of flint clay; in Georgia and California a distinct variety of flint clay is found in formations above the Carboniferous formations.

Ten or twelve years ago I visited several localities in West Virginia showing flint clays. In structure and analysis the clay resembled the Savage Mountain variety but no detailed examination was made to determine its geological position. That flint clays are found in West Virginia at other horizons than that of the Pottsville series is therefore not unusual.

* Received Mar. 9, 1916.

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The Newnam Hearth

BY WILLIAM E. NEWNAM,* B. S., COLLINSVILLE, ILL.

(New York Meeting, February, 1916)

THE smelting of galena in the ore hearth has been practiced in many countries for several hundred years with varying success. In the United States, the water-jacketed American hearths and the Jumbo hearths have found some favor in the Missouri lead belt, where large quantities of non-argentiferous galena concentrates are produced.

In form and method of operation the ore hearth has changed but little since its earliest conception. A detailed description of the furnace is not needed here; it has been fully described in most works on the metallurgy of lead and in many technical papers. Since its field is restricted to non-argentiferous galena, containing over 68 per cent. of lead, its use is limited to a few districts; and, on account of certain disadvantages attending its operation, modern sintering and blast-furnace practice has nearly driven it out of the Missouri field.

It is not generally known that the cost of producing pig lead is smaller by the hearth method than by sintering and blast-furnace smelting. Yet it is doubtful if any metallurgist knowing this to be true would recommend an American hearth installation—by reason of the disadvantages which will be briefly enumerated.

The furnace has been limited to a length of from 4 to 5 ft., requiring for its operation two men working an 8-hr. shift as a maximum. The product per man is small; the work is hot and laborious; and the difficulty of completely removing the dust and fume has made it a notorious source of lead poisoning. Although suitable for operations on a small scale a large number of hearths would be required for a plant smelting 10,000 tons of 68 to 70 per cent. concentrates per month; and the number of semi-skilled laborers would be six times the number of hearths—a condition which is prohibitive.

On account of the high percentage of dust and fume produced, a large flue and bag-house installation is required and, in the past, the handling and re-treatment of this dust and fume has been a serious problem.

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* Superintendent, St. Louis Smelting & Refining Co.

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On the other hand, the advantages are the immediate reduction of from 55 to 65 per cent. of the lead contents of the concentrates, with simultaneous roasting, thus producing a small tonnage of gray slag, which is an ideal product for the lead blast furnace, since it greatly accelerates the furnace operation and is productive of but little matte, the lead being largely present in the form of sulphate and oxide. As previously observed, the cost per ton of pig lead is somewhat cheaper by this method.[1]

Several attempts have been made to raise the tonnage and reduce the labor on the ore hearths by mechanical means, but up to the present without success. Although the furnace operation appears to be extremely simple, this is really not the case; and the ore hearth has persistently refused to respond to mechanical improvements. It is the purpose of this paper to describe a new and more fortunate endeavor in this direction.

After a careful study of the hand-hearth operation, it was decided that three things were necessary to perfect it, namely: A cool and sanitary hood; a mechanical rabbler which would effectually replace the continuous and laborious use of the hand rabble; and a lead-well attachment that would mold clean lead direct from the hearth basin with little attention on the part of the furnacemen.

All our experiments were carried out on the ordinary 4-ft. hand hearth. The hood problem was taken up first; and soon a double hood was devised which gave the furnace room a clear and fume-free atmosphere, at the same time reducing the direct and radiated heat to the workmen 50 per cent. A complete cessation of sickness and an increased output were at once noted.

After a few trials a simple and effective lead well and molding device was installed.

A traveling rabbler was next put on the 4-ft. furnace, and, after numerous changes, a successful type was worked out.

This rabbling machine, as shown in Figs. 1 and 2, is hung from a carriage which travels on an overhead track; and it rabbles in one direction only, for a reason given later. The machine being in such a position that the rabble arm is at the extreme end of the furnace, a releasing lever is pulled, which starts the machine, causing the rabble arm to describe a motion which is similar to that of the hand rabble, but more effective, since there is more power behind it. As the rabble arm is withdrawn from the fire, an eccentric connected with a ratchet wheel moves the carriage forward about 4 in. for the next stroke of the rabble arm. This motion is repeated until the other end of the furnace is

[1] It may be of interest at this point to note that the lead blast furnace carrying a charge with 20 per cent. of gray slag is smelting 7.5 tons of charge per square foot of tuyère area per 24 hr. The lead in this slag is under 1 per cent. and the proportion of fixed carbon consumed per charge 7.5 per cent.

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reached, when the machine automatically stops and withdraws the rabble arm from the fire.

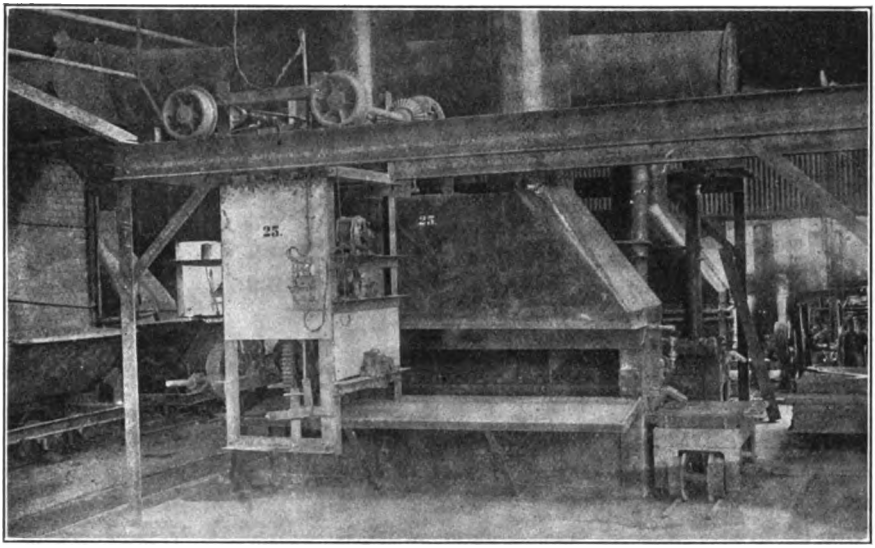

FIG. 1.—RABBLING MACHINE AT BEGINNING OF TRIP.

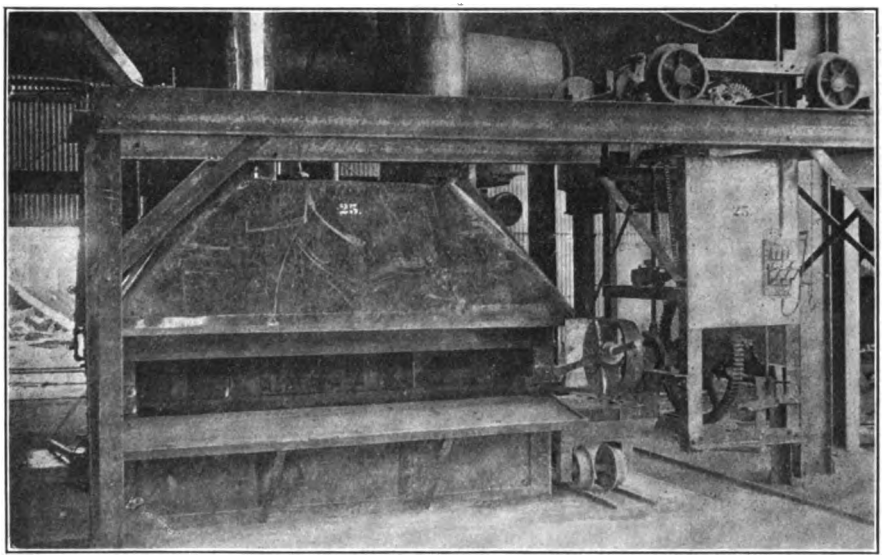

FIG. 2.—RABBLING MACHINE AT END OF TRIP.

Two men, a charger and a helper, operate the furnace. The helper follows close behind the rabble arm and with a long-handled shovel pushes back the loose charge, picking out the gray slag as he goes along. Close

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behind the helper comes the charger, who spreads a thin layer of ore on the charge as fast as it is shoveled back by the helper, adding coke breeze as needed.

Thus the fire is exposed for an instant only; and, by the time the trip down the furnace is completed, the end first charged is ready for the rabbling machine. It is for this reason that the rabbling is performed in one direction only. The first trip being complete and the gray slag removed from the apron, the throwing-in of a clutch causes the carriage to return without rabbling to its original position, where it automatically stops in readiness for a second rabbling trip.

Thus on each trip the fire is rabbled 24 times in 48 sec., the return motion requiring 12 sec. One horsepower is required under full load, and is supplied by a 1-hp., alternating-current, 220-volt motor. The rabbling machine is simple, strong, and durable, and only requires a few minutes' attention daily.

From the data compiled it was determined that the hearth could be increased to 8 ft. in length, that it would then produce 2.5 times as much pig lead as the hand hearth, and that two men could operate it with less fatigue than that sustained by two men on the ordinary hand hearth. An 8-ft. hearth was at once constructed, and from the first day of its operation it has not only realized our expectations but considerably exceeded them.

As direct-hearth labor is paid a certain rate per hundred pounds of pig lead produced, it will be seen that the cost of production as well as the number of furnaces and number of men employed has been more than cut in half. Moreover, the hearth laborers can earn more money per shift, through the increased efficiency per square foot of hearth area; and this, in turn, enables us to secure a more desirable class of men.

The following tables show that the 8-ft. furnace gives a much higher extraction of metallic lead, with a corresponding decrease in the amount of dust and fume produced.

Another gratifying feature is the reduction of the fuel added in the form of coke breeze. Whereas the hand hearth uses from 8 to 9 per cent. of fuel, the 8-ft. mechanical hearth consumes from 3 to 4 per cent. only. It is probable that the decrease in dust and fume as well as in fuel consumption is due to the shorter time the fire is exposed after rabbling on the 8-ft. hearth.

It has been found that by pugging the flue dust with a certain cheap chemical, and mixing it with a portion of burnt bag-house fume, the mixture can be successfully worked up on the hearth, giving a high lead extraction and a very low percentage of dust and fume.

Thus it will be seen that for galena concentrates of over 68 per cent. lead contents, the disadvantages of the old-style hearth have been overcome, and the cost of production has been lowered to such a degree that

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sintering with subsequent blast-furnace smelting is no longer comparable therewith.

Smelting Flue Dust on the Hearth

The following is an average of the results obtained by smelting a mixture of pugged flue dust and burnt bag-house fume. This mixture is made up so that the average weight of pig lead produced per shift will be approximately the same as that extracted from 70 per cent. galena concentrates.

Owing to the low percentage of sulphur present in the mixture as sulphides, 10 per cent. of coke breeze is required.

Burnt bag-house fume, containing 76 per cent., and pugged flue dust containing 62 per cent., are made into a mixture containing 67 per cent. of lead. Of this lead, 81.3 per cent. passes into pig lead, 10 per cent. into gray slag, and 8.7 per cent. into fume and dust.

The gray slag contains: Pb, 32.4; FeO, 14.7; CaO, 11.1; S, 1.2; and insoluble, 18.7 per cent.

The following table is a comparison of the two types of hearth, operating side by side on the same ore, showing the average results per 8-hr. shift over a period of four weeks, the labor being the same on each type.

Comparative Results

Ore charged, galena concentrates, containing 72.5 per cent. lead, and 15.1 sulphur.

	Newnam Hearth, Pounds	Hand Hearth, Pounds
Dry ore charged	13,179	5,091
Lead contents	9,554	3,691
Pig lead made	6,443	2,030
Gray slag made	3,318	1,329
Per cent. of coke breeze used	3.6	8.8
Rabble trips per hour	37.2	....

Percentage of Total Lead in Products

Pig lead	67.44	55.00
Gray slag	15.18	16.20
Dust and fume	17.38	28.80
	100.00	100.00

Analysis of Gray Slag

	Pb	FeO	CaO	S	Insoluble
Newnam hearth	43.7	12.8	9.9	1.9	12.6
Hand hearth	45.0	12.2	9.0	2.6	12.2

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The lead present is approximately combined as follows:

	Per Cent.
$PbSO_4$	10
PbS	6
PbO	25
Metallic lead	8.5

From the standpoint of roasting, the elimination of sulphur on the hearth is high in comparison with the other methods now in vogue.

Of the material drawn into the flues and bag chambers the proportion by weight is one of dust to two of burnt fume. From the analyses given below it will be seen that a considerable portion of the lead is present as sulphate, and on re-treatment the sulphate, reacting on the sulphide, greatly assists the desulphurizing process.

	Pb, Per Cent.	S, Per Cent.
Dust	62.0	11.1
Burnt fume	76.0	5.9

The lead present is combined approximately as follows:

	Dust, Per Cent.	Burnt Fume, Per Cent.
$PbSO_4$	18.9	53.6
PbS	55.4	1.2
PbO	2.0	41.5

Taking the above average run as a basis, the sulphur elimination is calculated as follows. The amount of fume and dust produced from the first dust and fume treatment is so small a percentage of the whole that it will not be considered here.

Newnam Hearth

	Pounds	Sulphur, Per Cent.	Sulphur, Pounds	Sulphur, Pounds
Dry ore charged	13,179	15.1	.....	1,990
Gray slag	3,318	1.9	63	.....
Dust	776	11.1	86	.....
Burnt fume	1,552	5.9	92	241
Pounds of sulphur eliminated	.....	.....	.....	1,749
Per cent. of sulphur eliminated	.....	.....	.....	87.9

Hand Hearth

	Pounds	Sulphur, Per Cent.	Sulphur, Pounds	Sulphur, Pounds
Dry ore charged	5,091	15.1	.....	769
Gray slag	1,329	2.6	35	.....
Dust	497	11.1	55	.....
Burnt fume	994	5.9	59	149
Pounds of sulphur eliminated	.....	.....	.....	620
Per cent. of sulphur eliminated	.....	.....	.....	80.6

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In several localities the ore hearth has been operated on high-grade galena, containing about 80 per cent. lead, with an extraction of from 80 to 85 per cent. of the metal. In order to determine the adaptability of this grade of ore to the Newnam hearth a test run was made, the average results of which, per 8-hr. shift, are given below.

Note the very high extraction of metallic lead and the correspondingly low amounts of dust, fume, and slag made.

Test on Galena Concentrates, Containing 82.0 *Per Cent. Lead and* 11.2 *Per Cent. Sulphur*

	Pounds
Dry ore charged	14,436
Lead contents	11,837
Pig lead made	10,790
Gray slag made	1,075
Per cent. of coke breeze used	2.4
Per cent. of crushed limestone	2.0

Percentage of Total Lead in Products

Pig lead	91.15
Gray slag	4.25
Dust and fume	4.60
	100.00

Sulphur Elimination

	Pounds	Sulphur, Per Cent.	Sulphur, Pounds	Sulphur, Pounds
Dry ore charged	14,436	11.2	.....	1,617
Gray slag	1,075	2.5	27	.....
Dust	254	11.1	28	.....
Burnt fume	508	5.9	30	85
Pounds of sulphur eliminated	.....	.....	.....	1,532
Per cent. of sulphur eliminated	.....	.....	.....	94.7

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Recent Advances in the Chemistry of the Cyanogen Compounds

BY J. E. CLENNELL,* B. S., OAKLAND, CAL.

(New York Meeting, February, 1916)

IT is a common observation that the improvements introduced in practice since the first announcement of the cyanide process have been almost entirely mechanical. Although a good deal of study and research has been devoted to the chemical problems involved, the results obtained appear trifling in comparison. Few of the suggested modifications in the chemical treatment have come into practical use and those only in limited fields. Nevertheless, if we study the recent history of the cyanogen compounds from a somewhat broader standpoint, we shall find that considerable additions have been made, not only to our knowledge of their chemical properties, but also to their applications for industrial purposes.

The present review of these advances naturally falls into two divisions: (1) processes involved in the treatment of ores; (2) manufacture of cyanides and other related compounds.

PART I. ADVANCES IN THE CHEMISTRY OF ORE TREATMENT BY CYANIDE

In the attempt to extend the scope of the process, various ores have been encountered which contain gold and silver in such forms that they will not yield satisfactorily to ordinary methods of cyanide treatment. In some cases the difficulty is due to the excessive action on the solution of base metals present in the ore; in others, to the occurrence of minerals such as dyscrasite (the native alloy of antimony and silver), or a supposed oxidized manganese compound containing silver, which are either altogether insoluble in or very slightly attacked by cyanide. A particularly troublesome case is that of the so-called "graphitic" ores, containing carbon in some form which has the property of precipitating gold and silver already dissolved.

Other efforts have been directed to improving extraction by electrolysis of working solutions, and by addition of various chemicals with or without electrolysis; to the diminution of cyanide consumption by "re-

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generating" the spent solution by chemical or electrical means with the object of decomposing complex cyanides formed in the treatment, thus reproducing simple cyanides or other substances capable of dissolving further quantities of precious metal.

Treatment of Carboniferous Ores and Products

Perhaps the most important recent papers bearing on the chemistry of ore treatment by cyanide are those of Morris Green[1] and W. R. Feldtmann.[2] (The references are to the bibliography at the end of this paper.) Green established the fact that the precipitation of gold by carbonaceous matter contained in certain ores and metallurgical products could be traced to the presence of occluded gases, of which carbon monoxide appears to be the most active. He also pointed out that graphite, anthracite and other dense forms of carbon, natural or artificial, have little or no precipitating effect. It was also found that charcoal gradually loses its precipitating power on exposure to air, and also loses it almost completely when heated to 500° C., if the occluded gases be extracted by a powerful vacuum pump. Feldtmann has carried this investigation a step further, and thrown light on the reaction by which gold is deposited from solutions of its double cyanides by carbonaceous matter.

An investigation was made on a West African ore containing graphitic schist, in the treatment of which the phenomenon not very happily termed "re-precipitation" had been a cause of poor extraction. Unsuccessful efforts were made to remedy the trouble by preliminary treatment of the ore with various metallic salts, oxidizing and reducing agents. Similar substances were added to the cyanide solution itself, but without effect. It was soon observed that an analogy existed between the effects of this graphitic schist, and those of charcoal on gold-bearing cyanide solutions, and that the action was entirely different from that of charcoal on gold chloride solutions. Gold deposited on charcoal from a chloride solution is visibly metallic, and readily soluble in cyanide, whereas gold precipitated by charcoal or graphitic schist from a cyanide solution is almost insoluble in fresh cyanide, and microscopic examination fails to reveal any metallic particles.

Finally it was discovered that a partial extraction of the precipitated gold could be made by treating the impregnated material with solutions of alkaline sulphides. Hydrosulphides of the type NaHS or KHS were the most effective; normal sulphides, as Na_2S, acted nearly as well; polysulphides were less effective and ammonium sulphide still less. It was shown that the gold so extracted appears in solution as an aurocyanide and not as an aurosulphide. It can be readily recovered by precipitation on metallic copper, and also, but apparently with more difficulty, by precipitation with zinc or aluminum.

Feldtmann offers the surmise that the gold compound formed both in

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charcoal and in the graphitic schist consists of a carbonyl aurocyanide, possibly $AuCN{\cdot}CO(CN)_2$, insoluble in cyanide but soluble in alkaline sulphides with formation of thiocyanates and aurocyanides, presumably by such a reaction as the following:

$$Na_2S + AuCN{\cdot}CO(CN)_2 = NaCNS + NaAu(CN)_2 + CO$$

An observation which may have an important bearing on the subject is that cyanogen gas is readily occluded by charcoal.

It is to be regretted that the investigation did not also cover the reactions of silver double cyanides under similar conditions. Ores are known in which gold is readily soluble in cyanide whereas the associated silver is extracted very imperfectly. Experiments of R. K. Cowles[3] indicate that the cyanide compounds of the two metals show some difference in their behavior in contact with carbonaceous matter. A selective precipitation was observed, first of the gold and later of the silver.

In the discussion on Feldtmann's paper[4] some alternative theories were put forward. S. J. Speak suggested that charcoal and the schist in question both probably contain oil, which may be the active precipitating agent. H. K. Picard thought the phenomenon might be due to oxidation, by a process similar to that occurring in the decolorization of organic matter by charcoal or platinum black, and stated that cyanates are always found in cyanide solutions after contact with carbonaceous matter of the kind in question. He suggested the reaction:

$$NaAu(CN)_2 + O = AuCN + NaCNO$$

for the precipitation, and the reaction:

$$3AuCN + Na_2S = NaCNS + NaAu(CN)_2 + Au_2$$

for the re-solution by alkaline sulphide, but admitted that it is difficult to see why, if the gold is precipitated as AuCN, it does not readily dissolve in cyanide. Moreover, if the precipitating effect is due to oxidation, it should be advantageous to pre-treat ores of this class with a strong reducing agent, but Feldtmann's experiments do not bear out this view. Picard, however, pointed out that it would be necessary, after such pretreatment, to prevent the carbonaceous matter from coming in contact with air before the application of cyanide solution.

Several speakers threw doubt on the actual occurrence of graphite in the West African schist. Trewartha-James doubted the theory of adsorbed gases and explained the phenomenon by "surface energy which produces precipitation by purely physical means." S. J. Truscott pointed out that in certain Westralian ores roasting did not completely eliminate the carbonaceous matter and that precipitation effects were still observed in the treatment of the roasted ore. He drew an analogy between this case and that of the manganiferous silver ores refractory to cyanide,

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in which pretreatment with a reducer has sometimes proved beneficial. F. P. Mennell thought that the association of this precipitation phenomenon with carbon was purely accidental and considered that other porous material might be expected to act similarly, the chemical composition of the porous medium having nothing to do with the effect.

Electrolytic Regeneration of Cyanide

Turning now to another field of investigation, we find that the process introduced a few years ago by J. C. Clancy has stimulated research on the effects of electrolysis on cyanide solutions, though as yet the results obtained have not yielded any important practical benefits in ore treatment. The Clancy process mainly depends on the electrolysis of water and the secondary reaction of the nascent oxygen thus produced on the thiocyanates present in working solutions, which are converted into cyanides by the reaction:

$$KCNS + 2KOH + 3O = KCN + K_2SO_4 + H_2O$$

Unfortunately, when the concentration of KCN in the solution is greater than that of KCNS, the former begins to be attacked, with the formation of cyanate, so that it is rarely possible, even under the most favorable circumstances, to "regenerate" to cyanide more than half the cyanogen which is present as a thiocyanate. Ferrocyanides are oxidized to some extent, by a similar reaction, but the conversion to cyanide is even less complete.

Other difficulties arise in connection with the material to be used as an anode in the process; all ordinary metals are rapidly corroded by the violent action of nascent oxygen at high current density. Peroxidized lead, "passive" iron, graphite, and fused magnetite were found to be the most resistant materials, though even these generally disintegrate in continued use.

In the course of some studies on the electrolysis of aqueous solutions of alkaline cyanides, E. F. Kern[5] points out that a consumption of cyanide occurs when aqueous solutions are electrolyzed by direct current with insoluble anodes, the consumption being due to oxidation. Anodes of iron, nickel, and lead are dissolved and immediately precipitated, lead as hydroxide and the other metals as mixtures of hydroxides and cyanogen compounds. The consumption of cyanide increases with diminished current density; at a high current density oxygen is evolved and less metal dissolved. Anodes of peroxidized lead and "passive" iron are more permanent than those of pure metals, and are not corroded unless exposed to the air, passive iron being superior to peroxidized lead.

In the electrolysis of cyanide solutions used for leaching refractory

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gold and silver ores containing sulphides, no reduced cyanide consumption and no increased extraction were observed, as compared with the treatment of the same ores without electrolysis. I have had a similar experience. When a low current density at anode and cathode was used, some active solvent was produced, but not enough to compensate for loss in electrolysis. The lower the current density at the anode and the higher at the cathode, the greater was the relative cyanide consumption.

When solutions containing thiocyanates or ferrocyanides were electrolyzed, the loss of cyanide was greatly diminished. Increasing alkalinity also reduced loss of cyanide by increasing conductivity. Peroxidized lead and passive iron anodes were corroded in presence of thiocyanates at current densities above 32 amperes per square meter.

G. H. Clevenger and M. L. Hall[6] also find that most of the decomposition of cyanide during electrolysis is due to oxygen liberated at the anode by decomposition of water. The final reaction results in the formation of carbonates, and when calcium salts are present, as is usually the case in solutions used for ore treatment, there is a precipitation of calcium carbonate. In some cases the latter might actually interfere with extraction by coating the ore particles.

On the whole, the results obtained by different investigators do not encourage the expectations entertained a few years ago that electrolysis will be of material benefit in the treatment of refractory ores by cyanide.

Effect of Mineral Ingredients in Water on Cyanide Consumption

Studies on this subject have recently been made by Thomas B. Stevens and W. S. Bradley[7], chiefly relating to the cyanicide effects of calcium and magnesium salts. They find that the common opinion with regard to the action of magnesium salts requires modification. It is known that these salts act as cyanicides by reactions such as

$$MgSO_4 + KCN + 2H_2O = Mg(OH)_2 + HCN + KHSO_4$$

and it is considered that $Mg(OH)_2$ is useless as protective alkali owing to its insolubility. These writers find, however, that its presence prevents any lime in solution being converted into $CaCO_3$ by atmospheric or dissolved CO_2, by reason of the reaction

$$Mg(OH)_2 + CO_2 = MgCO_3 + H_2O$$

The $Mg(OH)_2$ thus acts as a secondary protective alkali by allowing the lime to exert its full efficiency toward cyanicides present in the ore.

Another point noted by these writers, but not hitherto recognized, is the considerable solubility of calcium sulphate in cyanide solutions. Calcium sulphate and magnesium hydrate, though practically insoluble

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in water, are appreciably soluble in the presence of sodium chloride, a matter of importance in the case of the highly saline water sometimes used in Western Australia for making up solutions. It was found that the mixing of comparatively fresh water with solutions made from salt water, and consequently highly charged with calcium and magnesium salts dissolved in NaCl, results in the precipitation of $CaSO_4$ and $Mg(OH)_2$. These deposits are injurious, as they coat the zinc and cause bad precipitation of the precious metals; but they are soluble in strong cyanide or in ammonium chloride.

Various salts contained in the water used for making up solutions were found to act as cyanicides, the order of destructiveness being as follows: $Ca(HCO_3)_2$, $MgSO_4$, $CaSO_4$, $MgCl_2$, $CaCl_2$. The calcium bicarbonate acts much more rapidly than the other compounds. A test made by boiling the water in a reflux condenser, so as to expel CO_2 without reducing volume, and then filtering off the precipitated $CaCO_3$, showed that water thus treated had a greatly reduced cyanicide effect. When water containing calcium bicarbonate is used without previously being made alkaline or boiled, the reaction is as follows:

$$Ca(HCO_3)_2 + 2KCN = CaCO_3 + K_2CO_3 + 2HCN$$

the cyanide acting as a softening agent in a similar manner to lime; compare:

$$Ca(HCO_3)_2 + Ca(OH)_2 = 2CaCO_3 + 2H_2O$$

This reaction is the principal cause of the hardening of filter cloths, which occurs when a filter saturated with alkaline solution is washed with "fresh" water containing calcium salts.

The action of magnesium salts is much slower, and results in the gradual precipitation of $Mg(OH)_2$. Dissolved magnesium salts cannot co-exist with protective alkali even in the presence of much sodium chloride.

Part II. Advances in the Manufacture of Cyanogen Compounds

Fixation of Atmospheric Nitrogen

Much activity has been shown of late in the study of methods for the fixation of atmospheric nitrogen, and a number of products are now made on a large scale both for direct use and as a basis for the production of other nitrogenous substances. The most remarkable development has been in the manufacture of calcium cyanamide, $CaCN_2$. Impure commercial products consisting mainly of this substance are now extensively used as fertilizers under the names of "lime nitrogen" and "nitrolim." Alkali cyanides and other cyanogen compounds are obtained by further treatment of cyanamides in a variety of ways. Other

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processes are used to transform cyanamide into ammonia, ammonium salts, and other nitrogenous compounds of commercial value.

The earlier attempts at the fixation of atmospheric nitrogen were mostly based on the reactions taking place between barium oxide, carbon, and nitrogen at high temperatures. An important investigation of this process has been published by Thomas Ewan and Thomas Napier,[8] who summarize the chief data obtained by previous investigators in this field. The formation of cyanides from a mixture of BaO, C, and N requires a minimum temperature of 1,200° C., but proceeds best at 1,400° C., when 40 per cent. of the Ba may be cyanized. Hempel (1890) showed that the percentage of Ba cyanized may be increased by pressure. Readman (1894) applied the electric furnace for this process, which has been carried out on a large scale by the Scottish Cyanide Co. Frank and Caro showed (1898) that a large part of the nitrogen is fixed in the form of barium cyanamide, $BaCN_2$. Caro also found (1907) that the addition of alkali and alkaline earth fluorides accelerates the reaction, and enables it to proceed at temperatures of 900° to 1,100° C. Snodgrass (1907) showed that potassium carbonate has a similar action.

Ewan and Napier (1909) studied the action of catalysts to obtain the reaction at lower temperatures. In their first experiments, mixtures of barium carbonate and charcoal were placed in an iron boat in a porcelain tube, and heated in a current of dry nitrogen. The products finally obtained were barium cyanide $Ba(CN)_2$, barium cyanamide $BaCN_2$, carbon monoxide CO, and unaltered $BaCO_3$, C, and N. The following reactions are supposed to occur:

$$BaO + 2C + N_2 \rightleftarrows BaCN_2 + CO$$
$$BaO + 3C + N_2 \rightleftarrows Ba(CN)_2 + CO$$

They summarize their results as follows:

(1) Absorption of nitrogen begins at 900° to 930° C.

(2) The amount absorbed increases rapidly from 1 per cent. Ba combined at 930° to 40 per cent. at 1,000°, in tests made under the conditions that four molecules of N act on one molecule of $BaCO_3$ for a period of 2 hr.

(3) The greater part of the nitrogen is fixed as cyanide $Ba(CN)_2$ and not as cyanamide. When calcium is used in place of barium, the nitrogen is fixed almost exclusively as cyanamide, $CaCN_2$.

(4) At 960° about 2½ per cent. of the nitrogen is fixed; at 1,000° about 10 per cent.

(5) Addition of K_2CO_3 seems to improve the result, but the difference obtained falls within the limits of error in estimating temperatures.

(6) No cyanide is produced until about 30 per cent. of the $BaCO_3$ is converted into BaO, and until the percentage of CO has fallen to 30.

It is commonly supposed that barium carbide (BaC_2) is formed as

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an intermediate product, but Ewan and Napier find no evidence of this. From theoretical considerations BaC_2 could not be formed at all in presence of mixtures of CO and N from which N is freely absorbed by BaO and C. It is probable, however, that when calcium is used instead of barium the reaction:

$$CaC_2 + CO \rightleftarrows CaO + 3C$$

is a necessary intermediary.

Cyanid Gesellschaft[9] reports that nitrogen has no action on a mixture of CaO and C at 1,100°, and that the reaction $CaO + 2C + N_2 = CaCN_2 + CO$ takes place best at 2,000°. At this temperature nearly all the nitrogen is absorbed.

K. Kaiser[10] observes that if barium oxide or carbonate, intimately mixed with carbon, be heated to 900° to 1,400° C. and allowed to cool in a dry atmosphere of nitrogen under pressure, as much as 90 to 95 per cent. of the theoretical amount of nitrogen combines with the barium.

A French patent of 1913[11] describes a method of carrying out the reaction between barium carbonate, carbon, and nitrogen in an electric furnace. Tar is used as binding material for the ingredients; these are molded into balls and part of the tar recovered by distillation. The residue is then heated to 1,600° C. in the electric furnace and acted on by "producer gas" or other gas containing nitrogen, to form barium cyanide. The electrodes consist of two rings of carbon placed at the ends of the middle reaction zone of an inclined cylindrical furnace; the fused mass serves to conduct the current. Barium oxide may be regenerated by decomposing the barium cyanide by means of steam:

$$Ba(CN)_2 + 3H_2O = BaO + 2NH_3 + 2CO$$

When calcium is used instead of barium as the vehicle for the fixation of nitrogen, it was found by Barzano and Zanardo[12] that the yield of calcium cyanamide is increased by mixing calcium carbide with sufficient calcium cyanamide from a previous operation to form a friable mass, before subjecting it to the action of nitrogen.

The manufacture of cyanamide by the process of Frank and Caro is carried out on a large scale at the works of the American Cyanamid Co., Niagara Falls, Ont. Nitrogen is obtained by passing air over heated copper. The copper oxide thus formed is again reduced by the action of natural gas, so that it may be used repeatedly. Operations began in January, 1910, with a 10,000-ton plant, which has since been largely extended.

Cyanides and cyanamides of the alkali metals may be produced without the preliminary formation of a calcium or barium compound. E. A. Ashcroft[13] proposes to do this by using an alloy of the alkali metal with some heavy metal such as lead, and first fusing this with a portion of the

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product sought (*e.g.*, NaCN or Na_2CN_2) at about 600° C., to separate the alkali from the heavy metal. The resulting melt, freed from the heavy metal, is now caused to react with cyanogen or a substance yielding cyanogen, with or without electrolysis of the melt.

Conversion of Cyanamides into Cyanides, Etc.

We must now mention some of the recently proposed methods for converting calcium cyanamide into cyanides or other nitrogenous substances of commercial value. These may be roughly classified as fusion methods and wet methods.

Among the former, probably the best known is that of Erlwein and Frank[14] by which the crude calcium cyanamide is converted into a product containing the equivalent of 25 to 30 per cent. KCN, by fusing with carbon and common salt. Careful regulation of the temperature appears to be necessary. I have found that the process is not easily imitated in laboratory experiments. The product has been placed on the market under the name of "surrogat." For extraction of precious metals, it appears to be as effective, per unit of cyanogen contained, as ordinary sodium cyanide. It requires, however, to be digested with water and the insoluble carbonaceous residue filtered off before use.

J. C. Clancy[15] proposes to heat calcium cyanamide with its own weight of a mixture of equal parts of sodium sulphide and chloride in presence of carbonaceous matter.

E. E. Naef[16] treats crude calcium cyanamide at a temperature of 300° to 500° C. with sulphur or a substance yielding sulphur:

$$2CaCN_2 + S = CaS + Ca(CN)_2 + N_2$$

The product is then stirred with water and filtered from the residue of CaS + C. The filtrate is boiled gently with zinc dust, and the zinc cyanide formed converted into alkali cyanide by known methods.

Calcium cyanamide may be used as a source of ammonia by acting on it with steam under pressure at a temperature of 170° C., using an agitator to prevent formation of lumps. C. Manuelli[17] observes that under favorable conditions nearly theoretical yields of ammonia can be obtained at a cost of about 1.7c. per pound of nitrogen utilized. Another method (E. E. Naef)[18] is to pass a current of dry hydrogen, either alone or mixed with CO_2, CO, or N, at ordinary or higher pressure, over a heated mass of calcium cyanamide.

The wet methods are based on the fact that cyanamide compounds, such as $CaCN_2$, are transformed by water into the sparingly soluble crystallizable salt dicyandiamide $(CN{\cdot}NH_2)_2$. At a temperature of 13° to 14° C. about 90 per cent. of the nitrogen of calcium cyanamide may be dissolved in 25 times its weight of water, in 8 hr., the extraction proceed-

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ing rapidly at first, but very slowly later (C. Manuelli)[17]. Various salts of dicyandiamide may be prepared by digesting the substance in presence of acids with the oxides of the metals whose salts are required. The polymerization of cyanamide in aqueous solution is promoted by addition of fixed alkalies or ammonia. According to Grube and Kruger,[19] the reaction appears to be due to the union of undissociated cyanamide ($CN{\cdot}NH_2$) with cyanamide ions ($CN{\cdot}NH)'$, to form monobasic dicyandiamide ions $H{\cdot}(CNNH)'_2$, presumably in the following stages:

(*a*) Formation of cyanamide by hydrolysis of calcium cyanamide:

$$CaCN_2 + 2H_2O = Ca(OH)_2 + CN{\cdot}NH_2$$

(*b*) Dissociation of a part of the cyanamide:

$$CN{\cdot}NH_2 = H' + (CN{\cdot}NH)'$$

(*c*) Combination of dissociated with undissociated cyanamide:

$$(CN{\cdot}NH)' + CN{\cdot}NH_2 = H(CN{\cdot}NH)_2'$$

(*d*) Formation of dicyandiamide:

$$H(CN{\cdot}NH)_2' + H' = (CN{\cdot}NH_2)_2$$

It is found that at a given concentration of total cyanamide, polymerization proceeds most rapidly when the concentrations of cyanamide and of dissociated cyanamide ions are equal.

For the preparation of dicyandiamide from commercial calcium cyanamide, the addition of a foreign base is unnecessary; the conversion may be brought about by precipitating the lime at intervals so as to maintain approximately equal concentrations of $CN{\cdot}NH_2$ and $(CNNH)'$.

Alkali salts of dicyandiamide are produced by heating the substance with caustic alkalies in presence of an absorbent of water:[20]

$$4NaOH + (CNNH_2)_2 = (CN{\cdot}NNa_2)_2 + 4H_2O$$

If the water is not absorbed, part of it reacts with the melt thus:

$$Na_2CN_2 + 3H_2O = Na_2CO_3 + 2NH_3$$

The absorbents used are alkali or alkaline earth metals or their oxides, amides, nitrides, carbides or alloys.

Dicyandiamide is converted into cyanide by fusing with alkaline carbonates and carbon:

$$(CN{\cdot}NH_2)_2 + Na_2CO_3 + 2C = 2NaCN + NH_3 + N + H + 3CO$$

or it may be converted into a variety of nitrogenous products suitable for manures, explosives, etc.

The conversion of cyanamide into dicyandiamide may also be brought about by heating its solution with a catalytic agent. H.

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Immendorf and H. Kappen[21] propose to carry out this process by means of ferric oxide or hydrate, manganous or manganic oxides or their hydrates, stannic acid, chromic oxide or hydrate, hydrated silicic acid or similar substances. The products obtained are ammonia, urea, and dicyandiamide. During the process the catalyst absorbs the bases liberated and loses its efficiency. This may be restored by treatment with acid or with water. Commercial calcium cyanamide contains calcium chloride, which hinders the separation of urea and dicyandiamide. This difficulty may be avoided by converting the calcium chloride into sulphate or carbonate, which may be separated by reason of their insolubility in alcohol.

H. Kappen[22] obtains thiourea by treating dilute aqueous solutions of cyanamide with hydrogen sulphide at high temperatures, in closed vessels under pressure. The addition of acid and alkaline reagents and of soluble compounds of arsenic, antimony, or tin increases the efficiency of the process.

Production of Cyanogen Compounds from Nitrogenous Gases by Catalysis and Otherwise

Another method for the manufacture of cyanides and related compounds, which has been largely developed of late years, consists in the combination, under suitable conditions, of volatile carbon and nitrogen compounds. Usually this takes place at a high temperature under the influence of a "catalyst," *i.e.*, of some substance which does not undergo any permanent chemical change in the process.

It has long been known that hydrocyanic acid or ammonium cyanide could be produced by passing mixtures of carboniferous and nitrogenous gases over heated platinized pumice, as for example in Woltereck's process:[23]

$$CO + NH_3 = HCN + H_2O$$

C. Beindl[24] proposes to form hydrocyanic acid and cyanogen compounds by catalytic combination of gaseous and volatile carbon compounds with gaseous and volatile nitrogen compounds, by the use of a contact material consisting of a metallic oxide or a mixture of metallic oxides. The gaseous mixture should contain not more than 7 volumes of the carbon compound to 1 volume of the nitrogen compound. It is claimed that oxides of metals of the iron group have the property of inducing the rapid formation of hydrocyanic acid at relatively low temperatures, from dry mixtures of these gases.

It has been found[25] that the yield is increased by introducing into the arc of an electric furnace the vapors of metals and metallic compounds, such as those of copper or iron and their salts.

W. Moldenhauer and O. Wehrheim[26] describe a method by which

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nitric oxide can be obtained by catalytic combustion of cyanogen compounds. Ammonia may also be used as the source of nitrogen, but the yield of nitric oxide is less than with cyanogen, and more attention must be paid to the condition of the catalyst and time of contact with it.

J. E. Bucher[27] proposes to produce alkali cyanides by the action of atmospheric nitrogen on alkali carbonate and carbon dissolved in iron, the iron acting as a catalytic agent. The alkali cyanide formed is removed from the reaction zone by distillation under diminished pressure.

The Deutsche Gold und Silber Scheide Anstalt[28] propose to pass gaseous mixtures containing nitrogen obtained by heating certain waste products such as "vinasses" through highly heated conduits lined with refractory material such as Dinas brick, fused quartz, or a fused mixture of zirconia and quartz which remains gas-tight and non-porous at the high temperature employed. It is said that practically the whole of the nitrogen is obtained in the forms of HCN and NH_3, the relative proportions of these varying with the raw material used.

In the process of Barzano and Zanardo[29] a mixture of nitrogen (say 60 per cent.), hydrogen (32 per cent.), and hydrocarbons (6 per cent.) is raised to a high temperature, for example in the electric arc, and hydrocyanic acid absorbed by circulating the issuing gases through cooling and absorption apparatus. Sufficient nitrogen is added to maintain, with the hydrogen liberated, the concentration of these gases most favorable to the reaction.

Catalysis may be employed for the production of ammonia, not only from gaseous mixtures but also from solid organic nitrogenous compounds. F. Schreiber[30] proposes to act on the latter with contact masses containing hydrated iron oxide at temperatures below red heat. It is claimed that as much as 80 per cent. of the nitrogen in pyridine and in cyanogen salts may be thus converted into ammonia, while ordinary destructive distillation yields not more than 20 per cent.

Production of Cyanogen Compounds from Gaseous Mixtures by Absorption with a Metallic Salt

The cyanogen contained in crude illuminating gas has for some years been utilized as a source of cyanides and cyanogen compounds on a commercial scale, as in the methods of Rowland[31] and Bueb.[32]

L. Bergfeld[33] uses as purifying material dehydrated copper sulphate, or a mixture of salts, *e.g.*, alkaline earth sulphates or chlorides, which combine with NH_3 but not with H_2S, with metallic oxides capable of combining with H_2S. The mixture is treated with ammonia gas before use. The reaction is of the type:

$$H_2S + NH_3 + CaSO_4 \cdot NH_3 = CaS + (NH_4)_2SO_4$$

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At the same time some SO_3 (which is fixed by the purifying material) and sulphur are also formed, thus:

$$SO_3 + H_2S = H_2O + SO_2 + S$$

When the purifying material is exhausted it is heated, with regulated access of air, and ammonia liberated as follows:

$$CaS + (NH_4)_2SO_4 + 3O = CaSO_4 + SO_2 + H_2O + 2NH_3$$
$$SO_2 + 2NH_3 + 2H_2O = (NH_4)_2SO_3 \cdot H_2O$$

By gentle warming in presence of air, the ammonium sulphite deposited is rapidly oxidized to sulphate.

Cyanogen in the crude gas is fixed by the purifying material as cuprous ammonium cyanide. On heating with air this yields NH_3 and CO_2, any thiocyanate giving NH_3, CO_2 and SO_2. After passing the purifying material the coal gas may be further purified by leading it over alkali or alkaline earth sulphides, and if necessary over sodium amalgam.

C. A. Bergh[34] uses a solution of a zinc salt, containing also ferrous chloride, as an absorbent. The free acid formed is neutralized, and zinc sulphide and cyanide are precipitated, by addition of calcium carbonate, or some other sparingly soluble substance. This is added in quantity corresponding to the amount of H_2S and CN present. Iron remains in solution. By this method zinc is separated from iron and H_2S from CN by a single operation.

W. H. Coleman[35] passes coke-oven or other gases containing hydrocyanic acid through a tower packed with lumps of ferrous iron ore. A solution of sodium carbonate, or other suitable alkaline absorbent in solution or suspension, is circulated through the tower, and the resulting alkali or alkaline-earth ferrocyanide liquor is treated for recovery of the ferrocyanide.

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Segregation in Gold Bullion*

BY JAMES H. HANCE, E. M.,† IOWA CITY, IOWA

(New York Meeting, February, 1916)

INTRODUCTION

SEVERAL years ago the writer was connected with the Mint and Assay Service of the Federal Government as Assistant Assayer at the Salt Lake Assay Office. At that time cyanide bars formed approximately half of the bullion purchased by that office. Disagreement in valuation between the producer and the office was not infrequent and to a lesser extent between the Assay Office and the Mint, this latter issue being soon obviated, however, by adopting uniform methods of sampling the bars. A nice problem seemed to offer itself for research and the writer began a series of experiments which were soon terminated, however, by his going into other work, and which he regrets have not since been completed. The results attained may suggest lines of approach to this problem and are, therefore, offered in this article.

Frederic P. Dewey, Assayer of the Mint Service, U. S. Treasury, has had abundant opportunity to study this phase of metallurgy, and has published several articles relative thereto.[1] In the last article cited Dewey quotes analyses of cyanide bullion similar in content to that with which the writer worked, and which show some of the variations met with.

In the discussion of these papers,[2] comparison has been made in terms of copper bullion, but to the writer this seems of questionable value. Copper bullion may resemble silver bullion, but its similarity to gold bullion with little or no silver content is another thing. The freezing-point diagram of Roozeboom, reproduced on page 881 of the discussion cited, is for the gold-silver series as stated. The original problem under discussion, however, is *not this alloy* but gold with *base metals other than copper*. As for the statement that the corners are the worst places or

* This material in somewhat different form was submitted by the writer some years ago as thesis material in partial fulfillment of the requirements for an engineering degree from the College of Mines, University of Washington.

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[1] Frederic P. Dewey: The Assay and Valuation of Gold Bullion, *Trans.*, xl, 780 to 797 (1909); The Sampling of Gold Bullion, *Trans.*, xliv, 853 to 879 (1912); Cadmium and Nickel in Cyanide Bullion, *Engineering and Mining Journal*, vol. xciii, No. 15, 733–734 (April 13, 1912).

[2] *Trans.*, xliv, 879 to 882 (1912).

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positions to drill: In the writer's opinion there is not much choice, if the outer film of approximately 1 in. be excluded. A glance at the diagrams accompanying this article will show that a wide variation from the dip average is apt to be encountered at any of the places drilled in these experiments.

Sampling Methods

A few details of manipulation will be reviewed here to throw light on the discussion. The large bars are cast in molds 12¼ by 5¼ in., and range from 2 to 2½ in. in thickness. They are essentially prisms, the mold being tapered slightly to permit easy removal of the bar.

Three methods of sampling are in vogue: (1) Chipping diagonally opposite corners of the bar; (2) drilling the bar; (3) dip-sampling as the bar is being poured. Methods (2) and (3) are the ones commonly employed in sampling cyanide bullion. Drilling is done as shown in Fig. 1.

The two samples of top drillings are mixed and assayed as the top sample, and the two bottom drillings are mixed and assayed as the bottom sample. If the bars are not uniform in composition and if the variation is gradual it is readily seen that the top sample really represents an average of the top and center, or approximately a plane half way between the top and the center of the bar. Similarly the mixture of bottom drillings represents a plane half way between the center and the bottom. Now an average of these two planes is not necessarily an average of the whole bar, and where segregation is the *rule* and *not the exception*, this method may be open to criticism. This point, I believe, has been somewhat overlooked or neglected, and may well be worthy of experimental investigation. As emphasized by Mr. Dewey, these drillings themselves may be rather heterogeneous, the smaller fragments differing appreciably from the coarser in gold content, and this may result in wide discrepancies in assays of the same drill sample.

Dip samples may be taken as follows: One sample may be taken from the top of the melt just before it is poured into the mold, or a portion of the stream cut out just after pouring begins. This represents the top of the melt, or the bottom of the bar. A second sample taken from the pour just before completion represents the bottom of the melt, or the top of the bar. These samples are the most reliable, if the melt was thoroughly stirred just before pouring, and should check very closely within themselves and with each other.

Variations in Assays

Irregularities in the assay of cyanide bullion seem to have been first noted by Edward Mathey,[3] who thus summarizes some interesting

[3] Edward Mathey: On the Liquation of Certain Alloys of Gold, *Proceedings of the Royal Society of London*, vol. lx, pp. 21 to 35 (1896–97).

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and instructive experiments: "(*a*) Alloys of gold with base metals, notably with lead and zinc, now largely met with in industry, have the gold concentrated toward the center and lower portions, which renders it impossible to ascertain their true value with even an approximation to accuracy; (*b*) when silver is also present, these irregularities are greatly modified. The method of obtaining "cooling curves" of the alloys shows that the freezing points are very different where silver is present and where it is absent from the alloy; (*c*) this fact naturally leads to the belief that if the base metal present does not exceed 30 per cent., silver will dissolve it and form a uniform alloy with gold; (*d*) this conclusion is sustained by experiments (described in his article) which, in fact, gradually lead up to it and enable a question of much interest to be solved."

In another series of experiments with alloys of gold and platinum, Mathey clearly demonstrates the tendency of the latter metal to liquate toward the center of the mass while cooling.

Dewey's articles call attention to the existence of variations in assays of such bullion and the possible causes, but he does not show the nature of the variations.

Mention is made in metallurgical texts of the solvent property of silver where certain base metals, such as zinc, lead, etc., are present. The writer has had occasion to confirm this point in connection with the assay of gold bullion. If there is considerable silver present, 60 points (6 per cent.) or more, a uniform bar is generally obtained, but where the silver content is slight, segregation or liquation may take place on cooling and the resulting bar may be far from uniform in composition. With these points in mind the writer assayed a number of drillings taken from various portions of bars. Grouping these results in various ways a number of interesting and suggestive diagrams may be constructed which illustrate graphically the lack of homogeneity in some cyanide bullion.

Qualitative tests on some of the bullion indicated the presence of small amounts of mercury, lead, bismuth, arsenic, antimony, zinc and chromium. Dewey[4] reports cadmium, nickel, iron and copper from some similar bullion. The silver content ranged from zero plus up to 10 points (or 1 per cent.) in rare instances. As a rule these bars are very brittle, and upon fracture show some crystal faces well developed, and a variety of colors. These ranged from the color of marcasite and pyrite to those of chalcopyrite. The faces of the bars were characteristically marked with small gas blebs, giving them a pitted appearance. This was especially noticeable along the upper four edges.

Now when diagonally opposite corners of such a bar are chipped and assayed, the values obtained may have a wide variation among themselves, and in most cases are several points below the values shown by

[4] Frederic P. Dewey: Cadmium and Nickel in Cyanide Bullion, *Engineering and Mining Journal*, vol. xciii, No. 15, pp. 733 to 34 (Apr. 13, 1912).

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dip samples. Hence agreement between chip samples and dips from such bullion is hopeless, and the greater the number of duplicate assays made, the more firmly established is this difference. When this feature was recognized the mint issued the following instructions for sampling such bullion: "A drilling is to be made about one inch from each edge on two opposite corners of the top of the bar, and should extend half way through the bar. These two drillings are to be thoroughly mixed and assayed as the top sample. The bottom sample is to be taken in the same manner, corners being chosen not under the top corners drilled. (This is illustrated in Fig. 1.) Dip samples are to be taken as previously described and assayed with the drill samples."

In Table I are given some results obtained from assaying such samples. Where only one assay value is recorded, as for the top dips of bar 18,

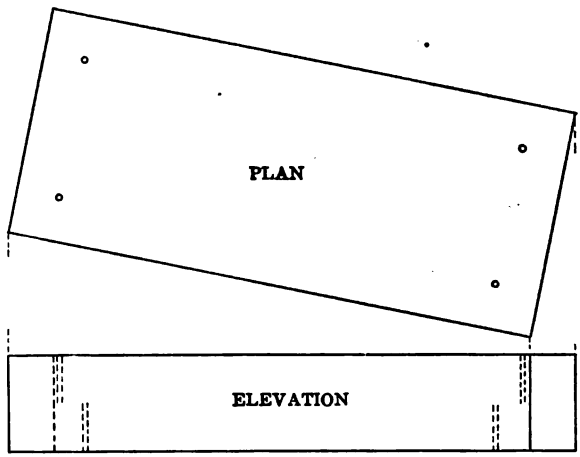

FIG. 1.

it should be understood that the other assay was spoiled, and not that it was too far off to put on record. Similarly for one of the top drills of bar 41. Eighteen of these bars were reported as having no silver (meaning less than 0.05 per cent.), the maximum silver content being 12 points (1.2 per cent.), whereas the average for the 48 bars was less than 3.8 points (0.38 per cent.).

Comparison of the assays of the top and bottom drills shows that in 38 cases the top drillings assay higher than the bottom drillings, whereas in 10 cases the reverse is true. In many instances a considerable range of assay values appears. Examining the dip assays in the same way it is found that in 33 cases the top dips assay higher than the bottom dips, and in 12 cases the reverse is true. In three bars the top and bottom dips average the same. As a rule, however, there is less difference be-

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TABLE I

Bar	Top Drills	Bottom Drills	Top Dips	Bottom Dips	Average of Drills	Average of Dips
1	865.1 865.2	865.9 866.6	867.4 867.0	864.2 864.2	865.7	865.7
2	845.4 844.4	845.5 845.3	843.4 845.1	845.7 844.9	845.15	844.78
3	835.5 835.4	834.5 833.7	834.1 834.7 834.5 834.6	834.8 835.3 834.8 834.1	834.78	834.63
4	857.7 856.8	856.6 855.5	859.5 859.2	859.6 859.5	856.65	859.45
5	856.9 856.7	855.1 855.8	857.0 857.4	857.0 857.2	856.13	857.15
6	842.9 843.3	844.8 845.0	844.3 844.2	843.7 843.7	844.0	843.98
7	795.7 796.1	790.0 789.9	792.0 792.1	791.6 791.9	792.93	791.9
8	836.2 835.7	834.1 834.2	834.1 834.1	833.5 833.7	835.05	833.85
9	775.8 776.4	773.8 774.4	773.5 773.6	773.6 773.2	775.1	773.48
10	810.1 809.8	806.7 812.4	810.4 811.1	810.7 810.8	809.75	810.75
11	793.6 792.7	792.0 794.7 793.1	793.2 793.3 793.2	793.1 792.5 793.4	793.22	793.17
12	795.2 795.3 796.0	796.1 796.5 792.7	795.3 795.2 794.9	793.8 794.7 795.9	795.3	794.97
13	799.4 799.2 796.9	796.9 797.1 802.8	798.7 798.6 797.7	798.0 798.0 798.5	797.05	798.25
14	928.4 928.5	928.0 928.1	929.5 929.1	928.9 929.0	928.25	929.13
15	834.2 834.1	831.4 832.2	832.8 833.0	832.3 832.4	832.98	832.63
16	854.5 854.7	853.2 853.5	853.5 853.4	853.3 853.5	853.98	853.43
17	849.4 849.2	847.5 847.8	847.6 847.6	847.3 847.5	848.48	847.5
18	898.4 898.3	898.6 897.6 898.5 897.5	898.9	898.8 898.4 897.6 898.0	898.15	897.94
19	788.3 788.1	787.9 787.8	788.8 789.0	788.9 789.0	788.03	788.93
20	803.7 803.6 802.5	802.8 803.7 802.9	803.4 803.1 802.0	802.7 802.6 803.2	803.2	802.87
21	796.4 794.5 793.7	794.3 794.0 796.0	794.6 795.2 794.9	795.1 794.8 794.4	794.82	794.87
22	875.1 875.4	874.7 874.4	876.0 875.9	874.9 875.3	874.9	875.53
23	813.3 813.2 813.2 813.1	813.1 813.0 813.3 813.9	814.5 814.3 813.8 813.4	814.0 813.5 814.5 814.7	813.26	814.09

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TABLE I.—(*Continued*)

Bar	Top Drills	Bottom Drills	Top Dips	Bottom Dips	Average of Drills	Average of Dips
24	813.1	812.7	813.5	812.8		
	812.3	812.8	813.1	812.8		
	812.5	812.2	813.3	813.4		
	812.9	812.3	812.7	813.4	812.6	813.13
25	806.4	806.8	807.3	806.9		
	806.8	806.3	807.2	806.8		
		806.2	.....	806.8		
		806.6	.....	806.6	806.52	806.93
26	875.3	873.9	874.8	874.3		
	875.0	873.4	875.0	874.2	874.4	874.58
27	772.7	773.7	774.1	773.7		
	772.8	773.5	774.3	773.6	773.18	773.93
28	811.6	807.8	806.8	808.1		
	810.4	807.7	808.9	807.9	808.88	807.93
29	772.2	769.6	771.9	771.1		
	772.1	770.3	772.1	770.9	771.05	771.5
30	758.1	755.5	757.8	756.7		
	757.9	754.7	757.8	756.6	756.55	757.25
31	743.6	743.6	742.7	741.6		
	743.0	743.3	742.6	741.7	743.38	742.15
32	875.6	874.3	875.7	875.2		
	875.5	874.2	875.4	875.0	874.9	875.33
33	886.8	886.7	887.8	886.5		
	887.2	886.6	886.7	886.6	886.83	886.9
34	860.3	858.7	858.6	858.2		
	859.6	858.5	858.5	858.4	859.25	858.43
35	868.2	868.2	868.1	868.0		
	868.8	868.4	868.0	868.0	868.4	868.03
36	885.7	884.2	884.1	884.0		
	885.1	884.1	884.0	884.0	884.78	884.03
37	887.0	885.5	885.3	885.5		
	886.8	885.6	885.4	885.7	886.23	885.48
38	890.2	888.8	889.2	889.2		
	889.5	888.7	889.1	889.1	889.3	889.15
39	888.5	889.5	889.7	889.9		
	888.7	889.5	889.9	889.7	889.05	889.8
40	896.1	895.6	895.9	895.7		
	896.1	895.5	896.0	895.7	895.83	895.83
41	903.8	901.0	901.8	901.4		
	903.6	901.1	898.7	901.5		
	903.6	901.0	902.0	901.9		
		901.2	901.8	901.8	902.2	901.36
42	892.3	888.5	888.7	888.8		
	890.4	888.6	888.9	888.7	889.95	888.78
43	893.8	892.5	892.0	892.8		
	894.8	892.9	891.0	892.7	893.5	892.13
44	939.8	938.9	938.7	938.7		
	940.2	939.1	938.9	938.8	939.5	938.78
45	935.1	933.7	933.5	933.3		
	935.2	933.1	933.5	933.3	934.28	933.4
46	942.0	941.2	941.1	941.3		
	941.7	941.5	941.2	941.3	941.6	941.23
47	694.3	704.0	698.4	699.6		
	694.3	703.9	699.5	699.4	699.13	699.23
48	854.2	853.0	851.2	853.0		
	853.8	852.8	847.8	852.8	853.45	851.2

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tween the top and bottom dips than there is between the top and bottom drillings.

Comparing the assays of drillings and dip samples it is seen that in 28 instances the drillings assay higher than the dips; in two cases they are alike, and in 18 cases the dips assay higher than the drillings. In nearly every instance the drillings show greater variation than do the dip samples, and yet in some samples (Nos. 1, 2, 12, 20, 28, 30, 31, 41, 43, 47, and 48) there is a marked variation here, even in dips from the same portion of the melt.

The molds are of cast iron and are heated to about 150° to 200°C. before the bullion is poured into them. As the metal is a better heat conductor than the air, that portion of the bullion which is in contact with the mold probably solidifies first, whereas the top cools last. This might produce a bar of the following characteristics: Since the four lower corners chill first, they might contain a higher percentage of the metals and non-metals of the highest freezing points. By the term "freezing point" as here used is meant a resultant of the absolute freezing point of the metal or alloy, and the saturation point of this same metal or alloy in the melt. This is offset to a certain degree by some of the metal solidifying as soon as it strikes the mold and not permitting any segregation. Next in order would be the four upper corners, then the bottom and sides, and lastly, the top which may be somewhat protected or blanketed by a thin coating of slag. The middle of each face would probably freeze later than any other point of that face, and the center of the bar, approximately, would solidify last, or possibly a thin film through the center and parallel to the bottom might be the last to take the solid form. The specific heats of the various elements making up the bar would exert a differential tendency in its cooling. If mercury were present, on account of its low melting point, it might tend to segregate to that portion of the bar which freezes last. The solvent properties of the metals and their alloying characteristics would also exert an appreciable influence here. Arsenic is known to cause a lack of uniformity, even where present in small amounts, but the exact nature of this action does not seem to be understood. A considerable amount of the volatile and easily oxidized elements are driven off during the melting stage, zinc, lead and bismuth being detected in the flues, but as charcoal covers are used, this expulsion is not complete. Unfortunately, an elaborate qualitative and quantitative analysis of this bullion was not feasible, owing to lack of equipment, and to certain restrictions placed upon the operating departments of the assay offices. Extensive drilling was therefore resorted to for a few bars, and some interesting variations determined.

From the drillings and dips taken in the usual manner for bar 49, the following assay values were obtained:

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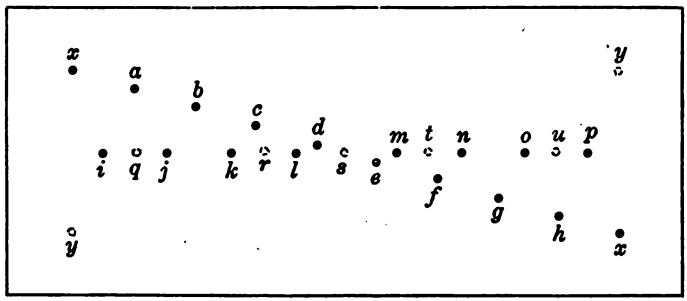
TABLE II

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
881.7 881.2	881.1 881.1	881.6 881.4	881.2 881.3	Less than one point of silver present.

This bar was specially drilled as indicated in Fig. 2, each drill hole extending to the middle of the bar. The bar was 10½ in. long, 4½ in. wide and 2½ in. thick, and weighed 824.81 troy ounces.

The points *x*, *x* represent the regular places for drilling the top of the bar, and *y*, *y* the places for taking bottom drillings. The points *a*, *b*, *c*, *d*,



Plan

1″
Scale

FIG. 2.

e, *f*, *g*, *h*, *i*, *j*, *k*, *l*, *m*, *n*, *o*, *p* are on top of the bar and spaced as shown. The points *q*, *r*, *s*, *t*, *u*, are on the bottom of the bar. The drillings from

TABLE III.—*Assay Values of Special Samples of Bar* 49

A	*B*	*C*	*D*	*E*	*F*	*G*	*H*
881.9 881.7	881.6 881.3	881.6 881.5	882.8 882.6	882.6 882.9	882.3 882.9	884.1 883.3	883.0 883.1

I	*J*	*K*	*L*	*M*	*N*	*O*	*P*
883.8 885.9	879.0 883.2	882.4 881.6	882.2 882.3	883.2 881.4	882.7 881.8	884.1 883.6	883.8 884.2

Q	*R*	*S*	*T*	*U*
885.7 884.4	884.7 882.2	880.9 880.1	880.5 881.4	882.5 882.3

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each hole were carefully mixed to insure uniformity (as far as possible) and were assayed in duplicate. Bars 50 and 51 were drilled in the same manner, and all the samples likewise assayed in duplicate. The values determined from these assays are given in the accompanying tables.

TABLE IV.—*Assay Values of Special Samples of Bar 50*

A	*B*	*C*	*D*	*E*	*F*	*G*	*H*
876.8 877.6	876.0 876.6	877.2 877.2	877.1 876.3	875.4 876.1	875.8 875.5	874.6 875.4	875.3 875.3

I	*J*	*K*	*L*	*M*	*N*	*O*	*P*
878.1 877.4	877.2 877.3	878.2 878.1	877.4 876.4	874.8 875.8	876.5 875.9	877.6 877.6	877.8 878.5

Q	*R*	*S*	*T*	*U*
877.4 878.5	875.6 875.5	874.0 873.2	875.1 875.5	876.6 877.1

From the drillings and dips taken in the usual manner for Bar 50, the assay values shown in Table V were obtained.

TABLE V.—*Assay Values for Bar 50, Samples Taken in Ordinary Manner*

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
874.3 874.0	874.4 874.4	874.1 874.4	874.2 874.4	Two points of silver present.

Bar weighed 822.09 oz. Dimensions same as Bar 49.

TABLE VI.—*Assay Values of Special Samples of Bar 51*

A	*B*	*C*	*D*	*E*	*F*	*G*	*H*
883.2 883.4	884.2 883.3	883.5 884.6	882.7 885.3	883.3 883.6	885.1 884.6	885.4 886.1	884.9 884.1

I	*J*	*K*	*L*	*M*	*N*	*O*	*P*
887.1 887.1	886.1 886.3	885.1 884.7	884.5 884.7	883.5 883.4	884.9 884.7	887.4 886.9	885.6 885.7

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TABLE VI.—(*Continued*)

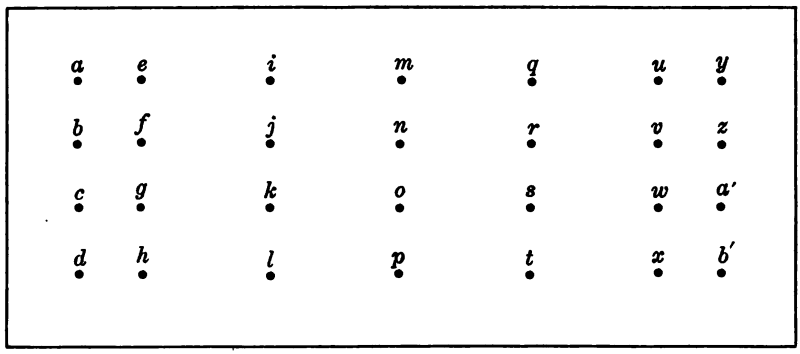
Q	*R*	*S*	*T*	*U*
888.7	883.9	883.1	883.7	884.6
888.7	882.8	882.3	883.1	885.7

From the drillings and dips taken in the usual manner for Bar 51, the assay values shown in Table VII were obtained.

TABLE VII.—*Assay Values for Bar* 51 *Samples Taken in Ordinary Manner*

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
883.0	882.2	882.2	882.8	Five points of silver present.
883.8	882.8	882.3	882.9	

Bar weighed 815.31 oz. Dimensions same as Bar 49.



Top Plan, Bar 52 Scale 1″

FIG. 3.

A large number of drillings were taken from two other bars. The method of drilling is shown in Fig. 3. The dimensions of the bars were: length, 12¼ in., width, 5¼ in., and thickness, 2¼ in. Bar 52 weighed 949.34 oz., and Bar 53 weighed 942.34 oz.

Drillings from *a* to *b′* through *z*, inclusive, were taken from the top of the bar. The bottom drillings *c′* and *d′* were taken under *y* and *d*, respectively. After assaying samples *a* and *b′* separately, the samples were carefully mixed and assayed as the top drills. Similarly, *c′* and *d′* were first assayed separately and then mixed and assayed as the bottom drills. Separate treatment of the end drillings shows different values for the two ends of the bar, a point lost sight of where the drillings are first mixed and then assayed. This becomes more noticeable when diagrams are constructed on these values.

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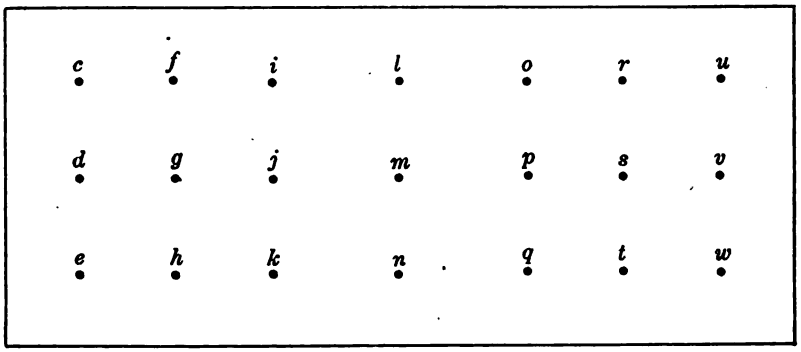
TABLE VIII.—*Assay Value of Special Samples of Bar 52*

A	*B*	*C*	*D*	*E*	*F*	*G*	*H*	*I*	*J*
838.2	840.8	839.9	839.4	838.1	840.0	840.8	838.9	838.4	838.7
838.0	840.5	839.6	839.1	838.0	840.3	840.4	836.7	837.6	838.7

K	*L*	*M*	*N*	*O*	*P*	*Q*	*R*	*S*	*T*
837.5	837.9	839.5	838.8	838.5	839.3	837.6	837.6	838.9	837.6
838.0	838.8	839.4	838.6	838.5	838.7	837.3	837.9	837.4	836.8

U	*V*	*W*	*X*	*Y*	*Z*	*A'*	*B'*	*C'*	*D'*
837.7	840.0	838.4	839.2	835.5	838.9	838.3	838.2	836.5	837.4
838.6	840.4	839.9	839.3	836.4	838.5	839.3	838.4	836.2	837.4

From the top drillings obtained by mixing samples *a* and *b'*, the bottom drillings obtained by mixing samples *c'* and *d'*, and the dips taken in the usual manner for Bar 52, the assay values given in Table IX were obtained.



Bottom Plan, Bar 53 Scale 1"

FIG. 4.

TABLE IX.—*Assay Values for Bar 52 Samples Taken in Ordinary Manner*

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
837.9	836.9	837.5	838.0	
838.2	836.8	838.2	837.6	Six points of silver present.
838.3	838.4	837.9	838.9	
837.8	838.4	838.4	838.8	

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Drillings *c* to *w* inclusive (Fig. 4), were taken from the bottom of the bar. Samples *a* and *b* were drilled from the top of the bar over *e* and *u* respectively. After assaying samples *a* and *b* separately, the samples were carefully mixed and assayed as the top drillings. Similarly, *c* and *w* were first assayed separately and then mixed and assayed as the bottom drillings. As was true with Bar 52, comparison of assays *a*, *b*, *c*, *e*, *u* and *w* shows considerable variation between the two ends of the bar.

TABLE X.—*Assay Values of Special Sample of Bar* 53

A	*B*	*C*	*D*	*E*	*F*	*G*	*H*
830.8	831.8	831.0	831.6	831.8	830.2	831.0	830.9
831.7	831.2	831.4	832.5	831.7	830.3	830.9	831.3

I	*J*	*K*	*L*	*M*	*N*	*O*	*P*
832.3	831.3	830.4	830.3	830.2	831.1	830.9	830.2
831.0	832.1	830.9	830.1	830.6	830.2	831.0	830.1

Q	*R*	*S*	*T*	*U*	*V*	*W*
829.3	829.3	833.2	829.0	829.9	831.5	828.8
829.7	829.4	832.8	828.8	829.0	831.1	828.7

From the top drillings obtained by making samples *a* and *b*, the bottom drillings obtained by mixing samples *c* and *w*, and the dips taken in the usual manner for Bar 53, the assay values given in Table XI were obtained.

TABLE XI.—*Assay Values for Bar* 53, *Samples Taken in Ordinary Manner*

Top Drills	Bottom Drills	Top Dips	Bottom Dips	
831.3	831.2	831.0	830.0	
830.5	831.4	831.1	829.7	Six points of silver present.
832.3	829.8	829.3	831.5	
.....	829.6	831.7	831.5	

Construction of Diagrams

The horizontal length of the diagram in each case represents the outside dimension of the bar along the line of the profile. As no sample is taken less than an inch from the edge of the bar, the curves all terminate inside of the diagrams. The vertical control is that of fineness, the

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extremes taken so as to include all assay values on the bar. This feature is apparent on the last or composite diagram for each bar, upon which are assembled the assay values of all the *drillings* from that bar. In this diagram the values are all projected upon a plane passing lengthwise through the center of the bar and normal to the end and bottom faces.

The light solid horizontal line passing through the diagram—and curves in *some cases*—represents the average value of the dip assays, and probably is a very close approach to the true value of the bar. The

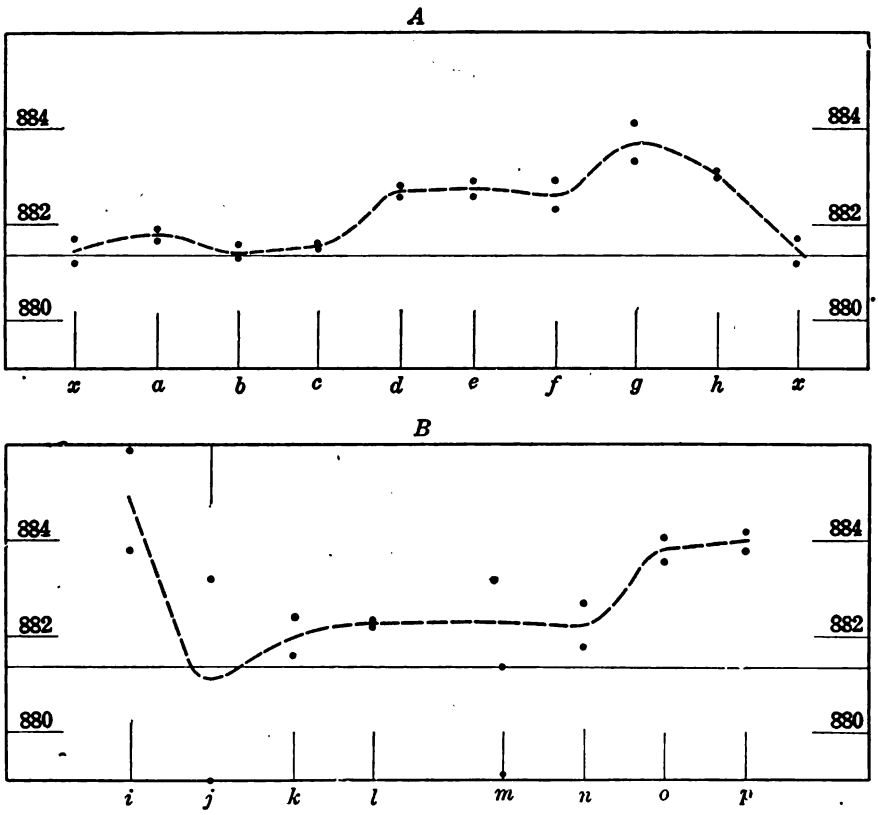

PLATE I.—BAR 49.

letters along the base of the diagram are the points on the bar from which the drillings were taken, and over these letters are placed the corresponding assay values. Each diagram is thus an assay profile along a line through the lettered points, the last diagram for each bar being a composite profile along a longitudinal medial plane, all assay values of the *drillings* being projected upon this plane. *All drilling* assay values are placed upon the diagrams except Plates XII and XIII, and the curves are drawn through the average for any point.

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Several methods of drawing curves through these average values are possible, and each one may be nearly equally justified.

1. Straight lines may be drawn between consecutive average values, the profile consisting of a zigzag line made up of straight segments.

2. A smooth curve may be drawn through consecutive average values, without regard to what might lie beyond each end of the curve.

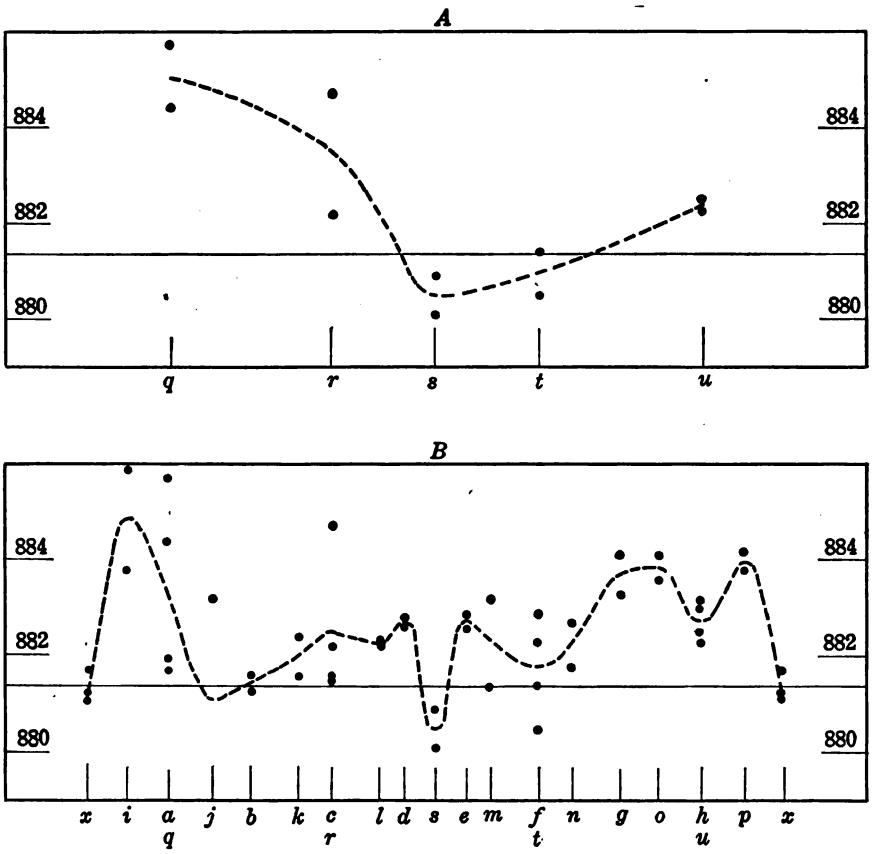

PLATE II.—BAR 49.

3. A smooth curve may be drawn through consecutive average values, and this curve be bent toward the horizontal at each end.

4. A smooth curve may be drawn through consecutive average values, but warped toward the average dip value. This would assume all other than dip values as departures from the true value and as abnormalities.

5. An average of two consecutive averages may be taken and its departure from the dip average noted (call this departure *d*). Now take the dip average as a datum line, and half way between the two averages considered (horizontally), insert *d* with its sign changed. This is merely

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a device for obtaining low values to offset the high values. Either short straight lines or a smooth curve may then be drawn through all points, average values and the new points thus computed.

The first method assumes that assay values were taken at all maximum and minimum points, and that the gradation between points is uniform. This is certainly not necessarily true, and is probably never attained.

The second method considers that all maximum and minimum points are represented, but does not suggest uniform gradation between points sampled. Thus the curve between points is dependent upon adjacent

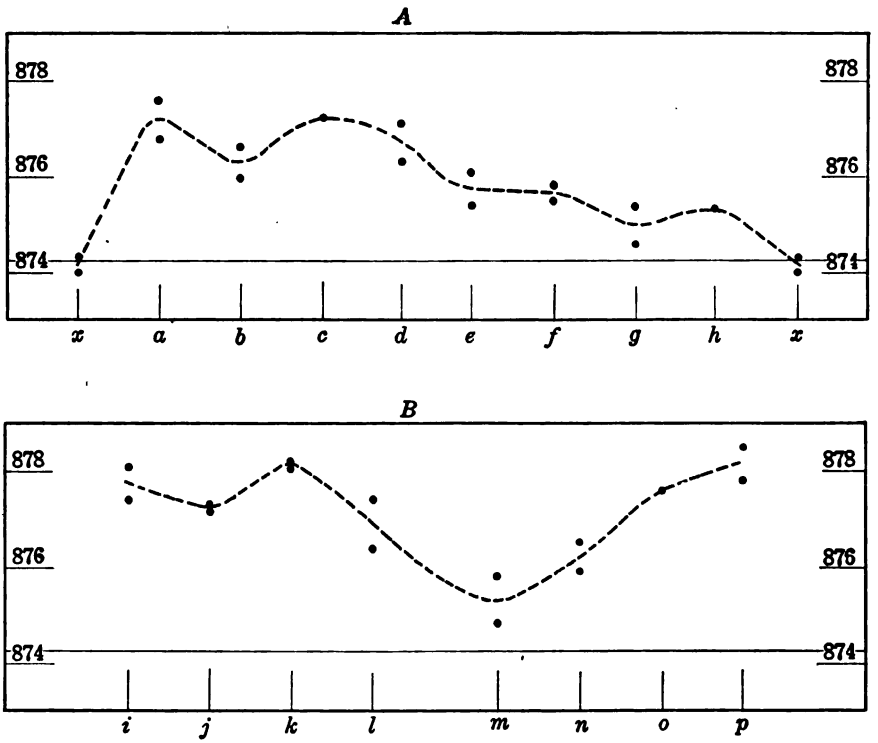

PLATE III.—BAR 50.

values as well, and therefore is probably in less error than the zigzag line of method (1). As it is not necessarily true that all maxima and minima are represented, there is undoubtedly a departure from actual conditions, but the only remedy for this would be a determination of essentially all points of the bar, a proceeding manifestly impossible.

The third method bends the curve toward the horizontal at each end. This undoubtedly possesses merit in some cases, but may introduce errors in other instances. It is essentially the same as (2).

The fourth method brings out interesting contrasts, but certainly

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cannot represent true conditions in the bar. It might still leave all or most of the profile above or below the dip average, a lack of balance probably not the case. A glance at the assay values and diagrams for bars 49, 50, and 51 will show the difficulty of realizing such conditions with some bars.

Such a diagram as outlined in the fifth case or method would show some of the variations demanded if we are to consider the dip average as the true value. Although introducing mid-points, in the opinion of the writer it may more nearly meet the conditions of the case than any of the other

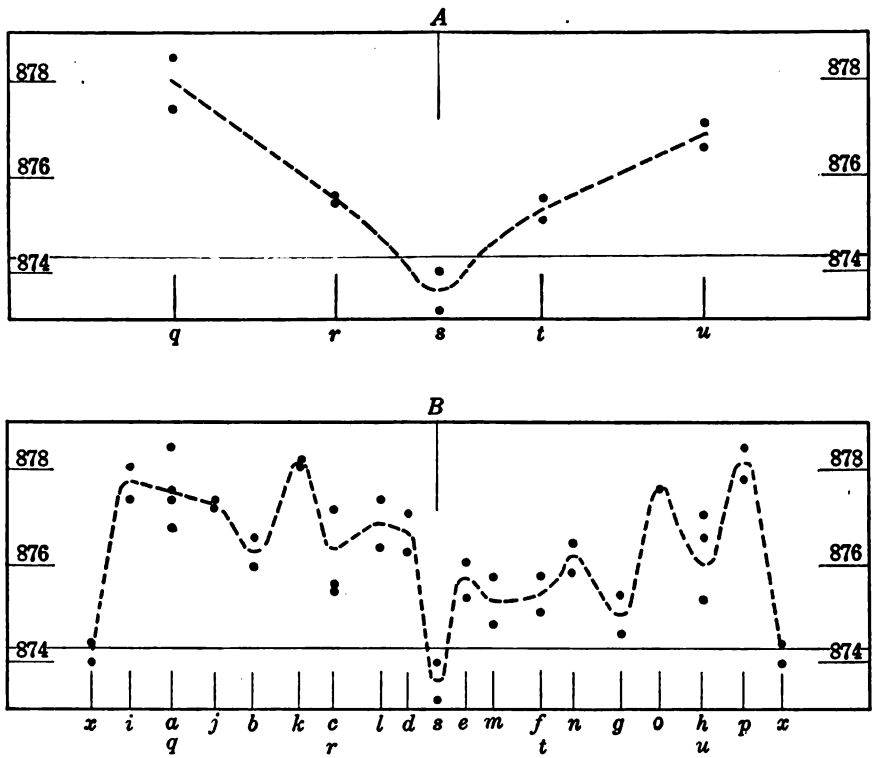

PLATE IV.—BAR 50.

methods. In all probability it does not properly place these variations, but at the same time it shows what the minimum variations must be if the dip value is the true datum line. For instance, such a composite diagram as Plate VI. *B*, cannot tell all the story, for where are the low values to offset the high ones and give the true bar value? Much of this adjustment is undoubtedly accomplished in the outer film of the bar, outside the curve terminals, but not necessarily all.

In the first nine plates the second method was used. For bar 53 the third method was followed, and it shows curves nearly identical with

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those obtained with the second method. In Plate XIII the first and fifth methods were used.

Discussion of Curves

Bars 49, 50 *and* 51.—As these three bars were of the same size and shape, and as they were sampled in identically the same manner, a comparison of their diagrams may be instructive.

Noting the curves along the diagonal line on top of the bar, it is seen

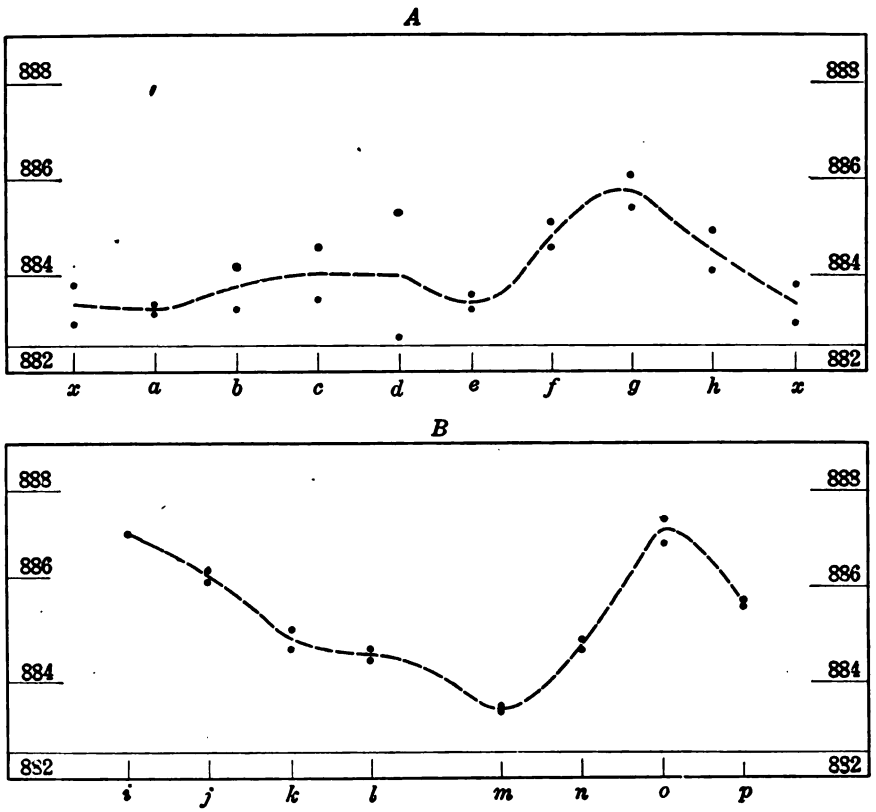

PLATE V.—BAR 51.

that essentially all the *average* values are above the average dip value, emphasizing the difficulty of obtaining the fineness of such bullion from drillings. Another feature is the large bulge or wave-like crest in the curve near one end. The reason for this is not known, as the melt is poured across the mold from end to end, and principally at the center. As the drillings from each end are mixed and assayed in duplicate, both end values are alike. Duplicate values would not likely result if the drillings from each end were assayed separately. In the case of each of these three bars the curve is convex upward.

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Examination of the curves for assay values along the central line on top of the bar shows a similar wide range of values and a still greater departure from the average dip value. Each of these curves is essentially concave upward, in contrast to the convexity of the curves along the diagonal line. In speaking of the cooling of the melt, mention was made of the probability that the center of the top of the bar may be the last part of this face to solidify. If this is true in principle for the bar itself,

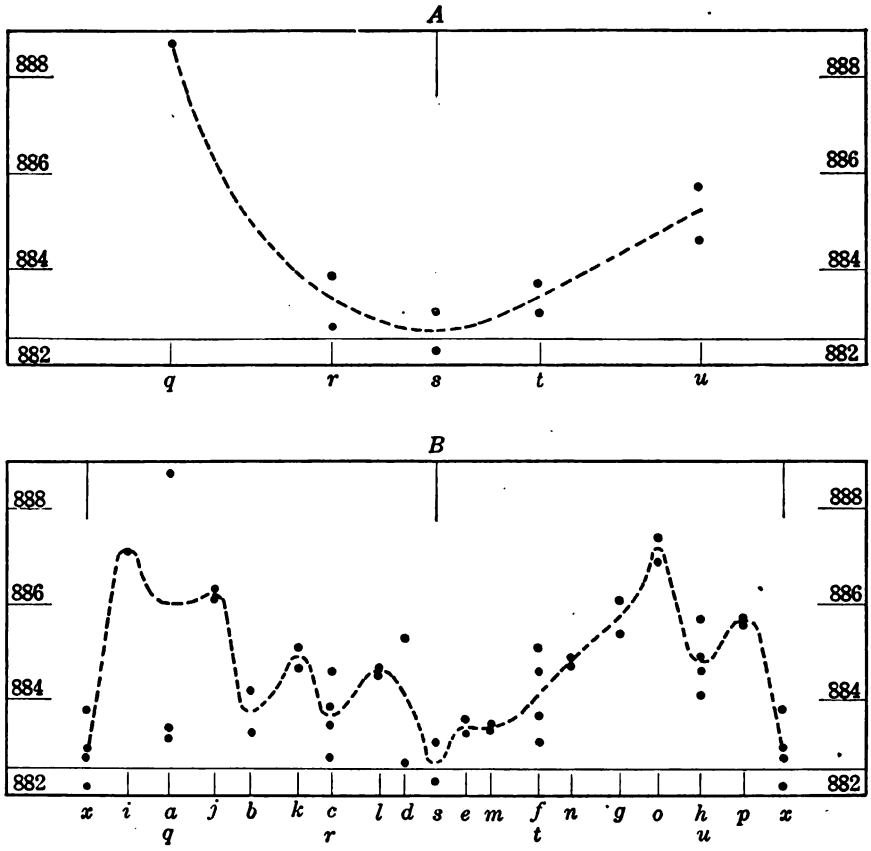

PLATE VI.—BAR 51.

and a metal such as mercury were present, a lower assay value for this portion of the bar might be expected. However, it might be expected to fall below the average dip value rather than above. As in the previous case, nearly all the assay values lie *above* the average dip value.

The curves for the central line along the bottom of the bar have one noticeable trait in common; namely, a concavity upward which is due to a lower value near the center of the bar. In keeping with this feature is the hypothecated action due to the unequal rates in cooling. The

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center of any face probably cools or solidifies later than any other portion near the edges, such as along the borders or near the corners. At the centers, these curves, except that on Plate VII, *A*, pass below the average dip value, although most of the curve is above.

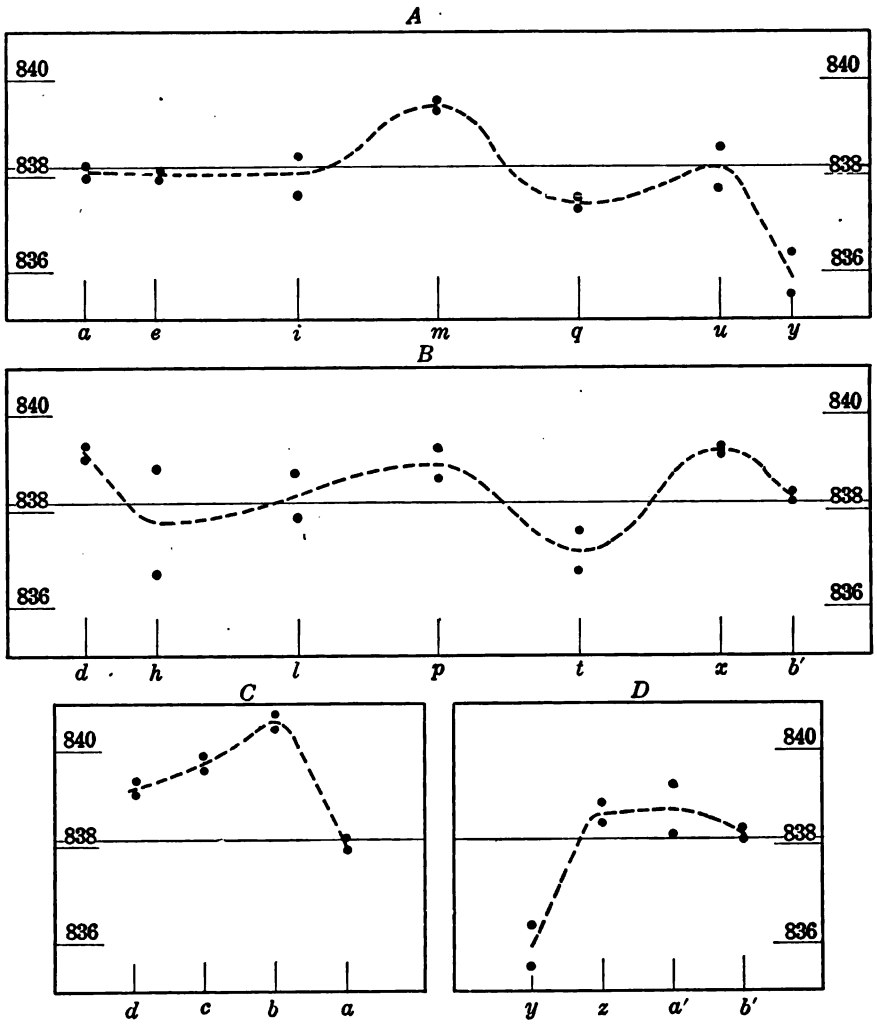

PLATE VII.—Bar 52.

In the composite diagrams the curves are still more irregular but all illustrate the high values of drillings and the inadvisability of attempting to determine fineness of such bullion from such samples.

In Plates II, *B*, and IV, *B*, the end averages are nearly coincident with the dip averages. In Plates VI, *B*, IX, *D*, and XII, *B*, however, these end

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average values are not the same as the dip averages, but are higher in some cases and lower in others.

Bars 52 and 53 were sampled according to a different scheme. Lines were drawn parallel to the edges, and drillings were taken at short intervals along these lines (see Figs. 3 and 4).

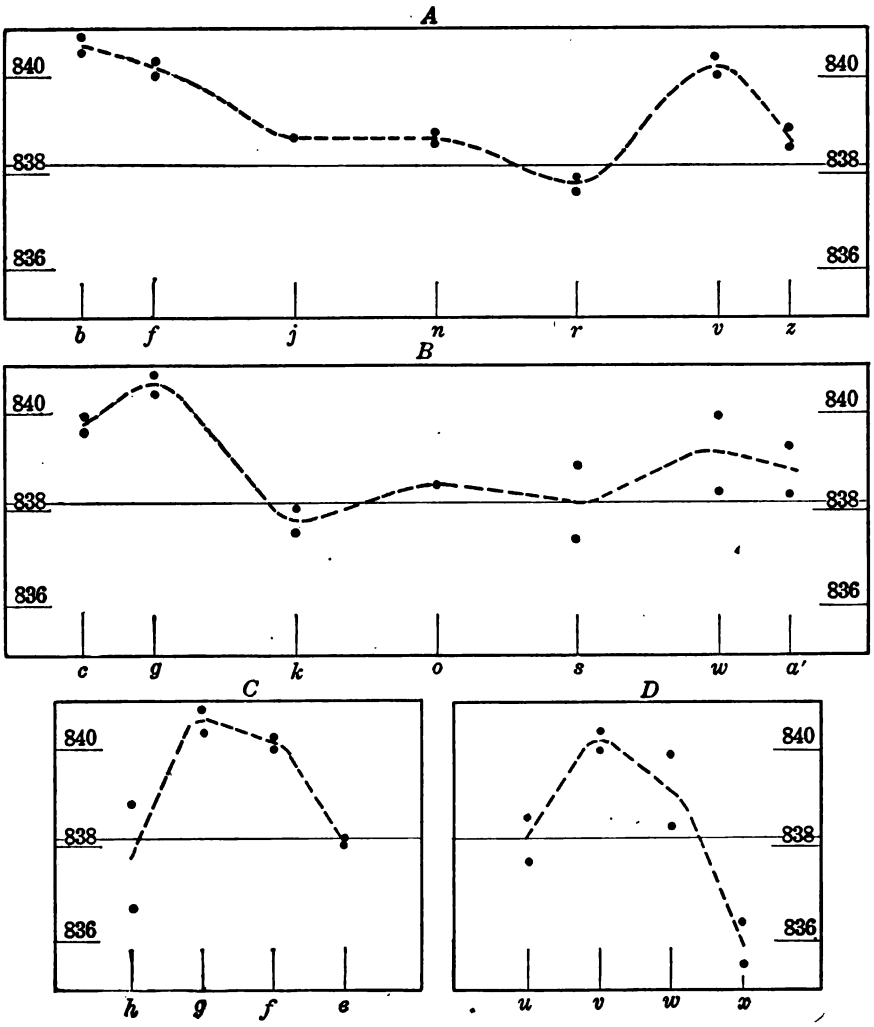

PLATE VIII.—BAR 53.

The bottom of bar 52 and the top of bar 53 were sampled in the official manner previously described. Plotting these values lengthwise and crosswise of the bar, two sets of curves were obtained. A composite diagram is also given for each bar. The diagrams are arranged in pairs so as to contrast similar portions. Thus for bar 52 the profile along

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aeimquy corresponds in position to that along *dhlptxb'*. Each curve connects a series of averages for assays of drillings taken parallel to, and at a given distance from the edge of the bar. On a given face these profiles

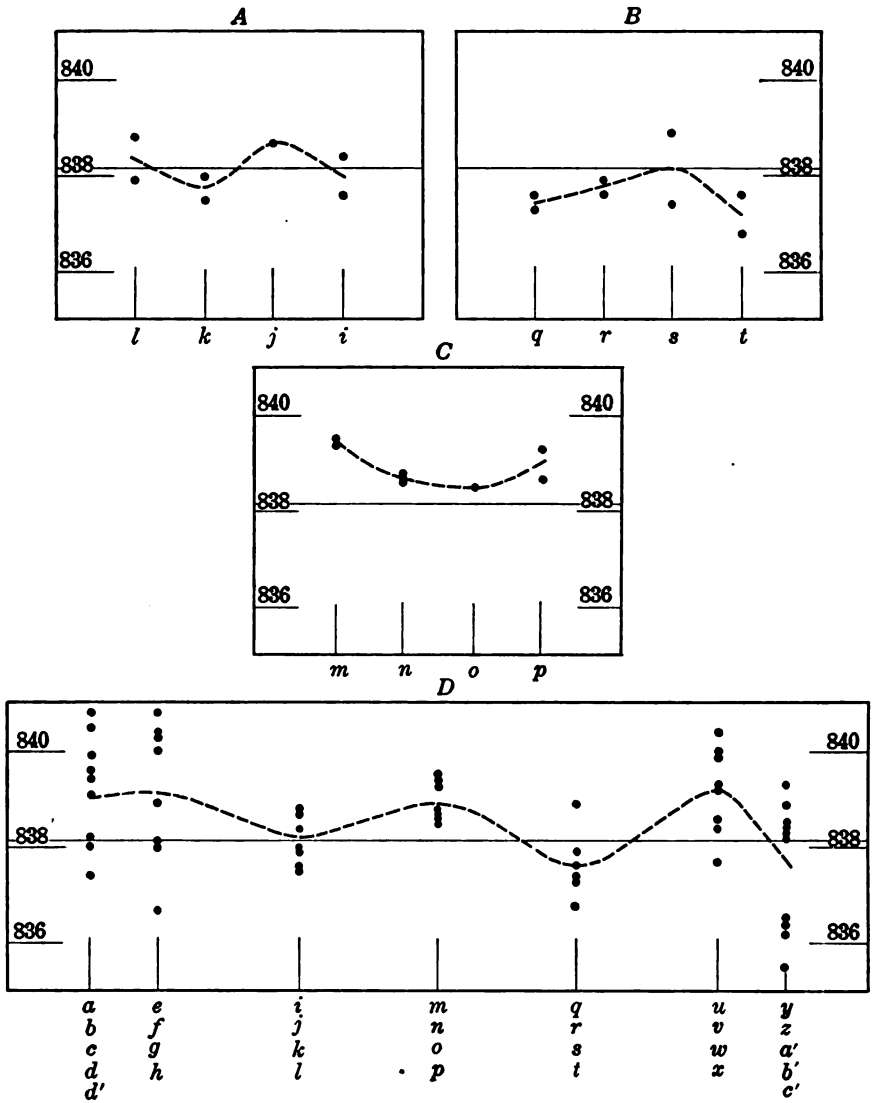

PLATE IX.—BAR 52.

might be expected to show similar variations if the liquation in such cooling is merely a matter of rate of cooling rather than a complex function depending upon many variables, conspicuous among which is apparently composition. Possibly the term liquation is here inapplicable, as the

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solid bar may (and in all likelihood does) somewhat resemble a rock of heterogeneous mineral composition, rather than a liquated magma. Close proximity to the walls of the mold may exercise much the same function as proximity to inclosing wall rock in the case of a cooling magma. Here, however, the solution of metal in metal (or non-metal)

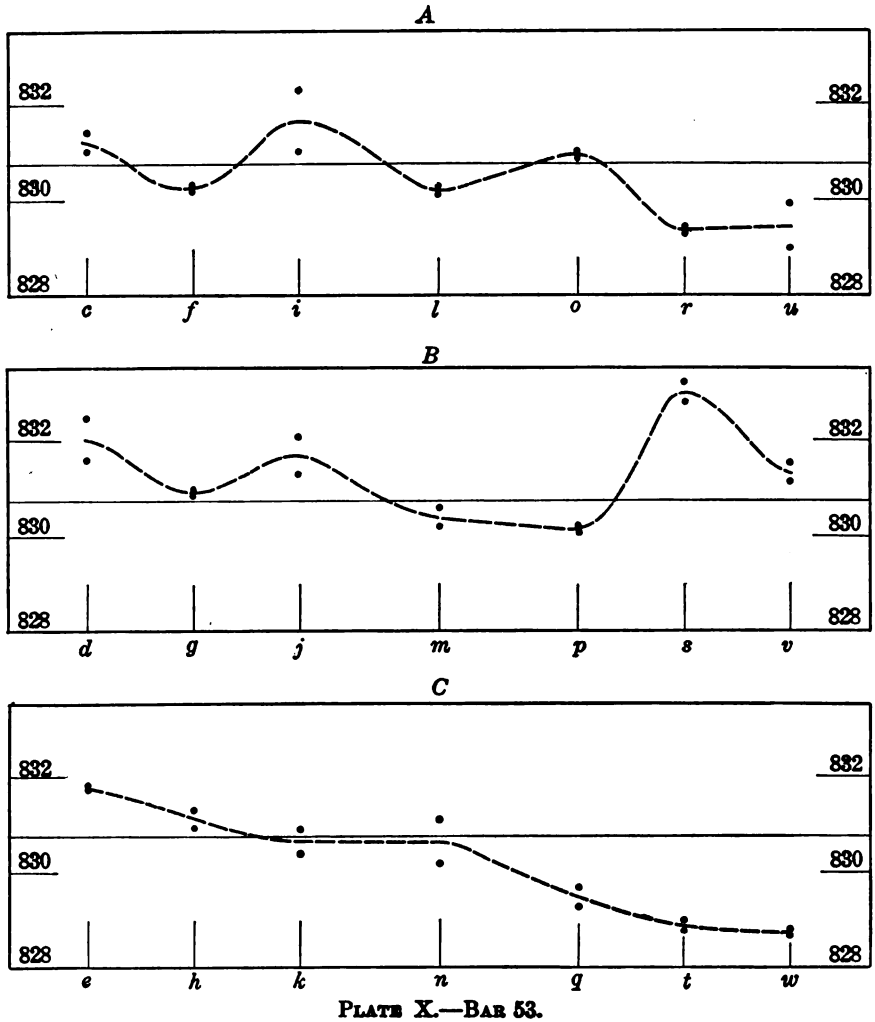

PLATE X.—BAR 53.

and the peculiar properties of alloys may vary the final products considerably.

Thus in Plate VII, *A* and *B*, each curve shows a conspicuous crest the the center and a conspicuous trough to the rig[illegible] rage. The [illegible] crest [illegible] of the right half of the curve shows two crests with [illegible] minima points.

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each case the central crest separates two dissimilar curves. Neither curve is decidedly concave or convex upward. Unlike preceding curves they are approximately bisected by the dip average.

Comparing *C* and *D* a slight similarity is noted, in that both are convex upward and the variation from one edge of the bar to the opposite

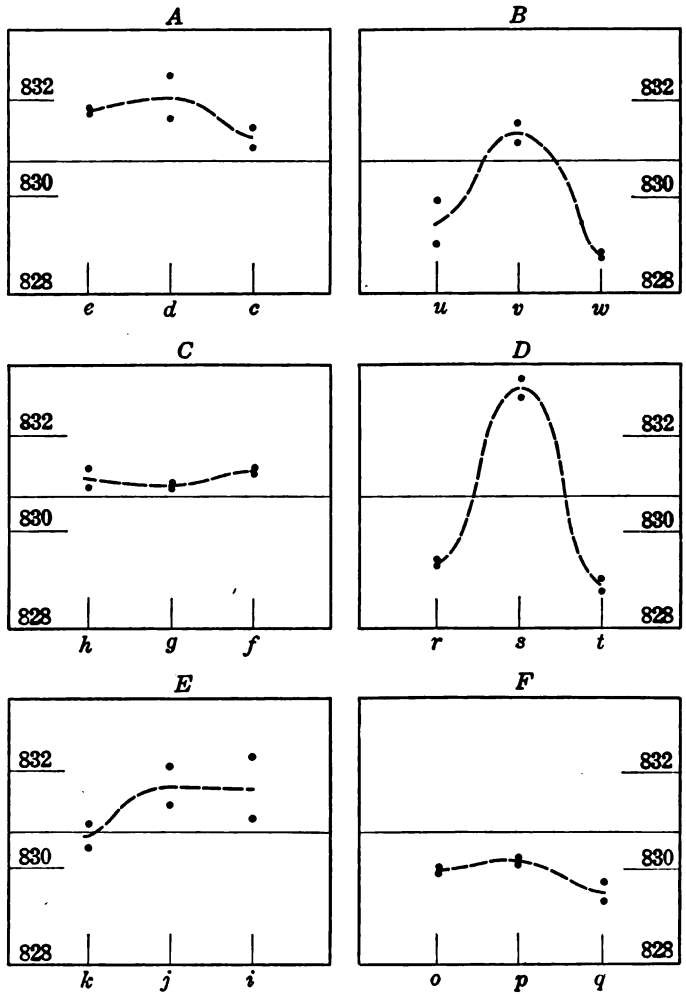

PLATE XI.—BAR 53.

side is consistent for both profiles. Considerable variation, however, is shown in these profiles.

In Plate VIII, contrasting *A* and *B*, some like features are apparent. The curves are both concave upward and average well above the dips. depending on many which reaches nearly the same minimum value, composition. Possibly the ar maxima.

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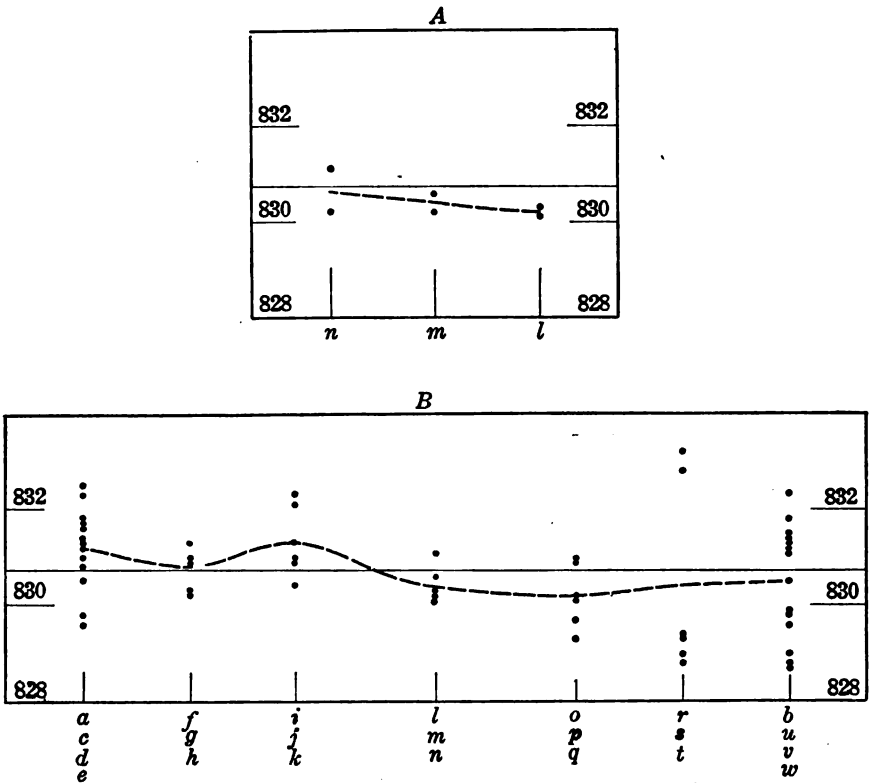
C and *D* are both convex upward and average well above the dips. These curves which are convex upward, are just the opposite of *A* and *B* which are concave upward, and show that if there is such a thing as concentration of a metal such as mercury toward the center, it is more than offset by some reverse process, at least locally.

In Plate IX, *A* and *B* are not much alike. *A* is unlike *C* on Plate VIII, although it is an adjoining profile, whereas *B* is similar to *D* of Plate VIII, but less pronounced. *C*, Plate IX, is concave upward and



Mixed Top and Bottom Drills not Included on Composite Diagram

PLATE XII.—BAR 53.

in harmony with a possible concentration of such a metal as mercury at the center. At variance with this, however, is the fact that the entire curve is above the dip average.

The composite diagram *D* shows a curve with three well-defined crests and two troughs, and with an average somewhat above the dip average. Although the range in individual values is considerable, the point averages are fairly close to the dip average. The grouping of the assay values shows well-defined maxima and minima points.

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Bar 53 was drilled in a way somewhat similar to bar 52 except that the drillings were from the bottom of the bar instead of the top. This, it was hoped, might furnish data for contrasting cooling in a face exposed to the air or to a thin slag coating, as compared with the cooling of a face next to the mold. Drilling one or several of the lateral faces might show other gradations and bring out interesting contrasts.

In Plate X, *A* and *C* are similarly situated, and both show downward

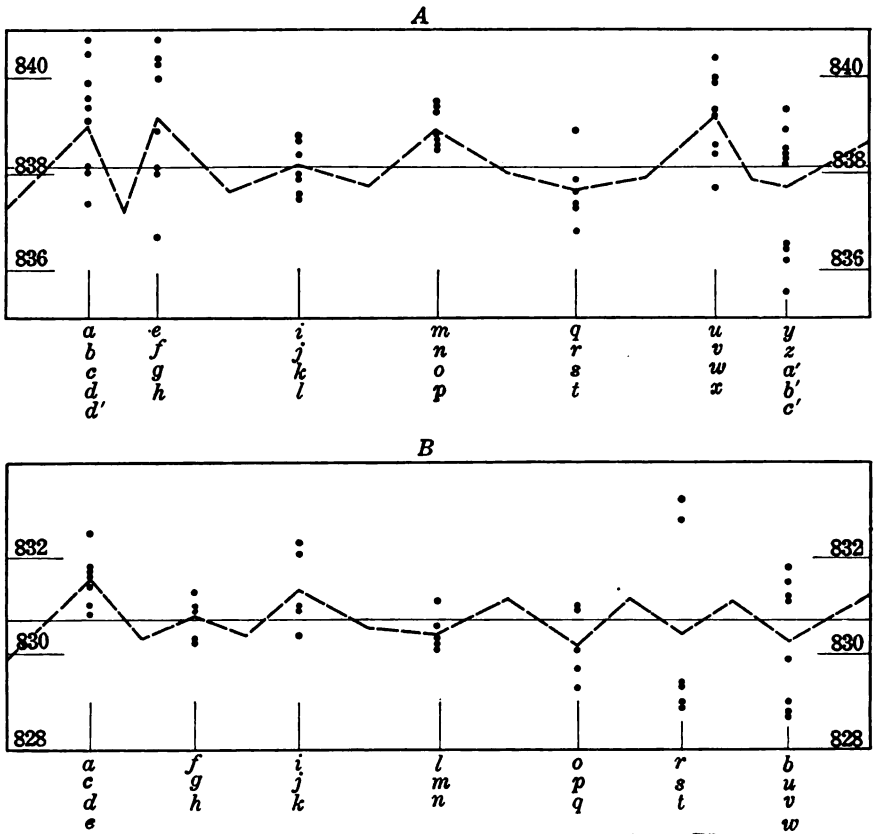

PLATE XIII.—BARS 52 and 53.

gradients from left to right. Neither one, however, is decidedly concave or convex upward, and the curves are not very similar. Curve *A* shows erratic tendencies whereas *C* indicates a less variable decline in value from left to right. *B*, which is along the medial line of the bottom of the bar, is concave upward, but is in decided contrast with both *A* and *B*. In some ways, however, it seems to be a composite of both *A* and *C*, except for the values of *s* and *v* which seem abnormally high for this bar.

In Plate XI most of the curves are convex upward, but do not go

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together well in pairs. For instance, *A* and *B* are similarly situated, but are in strong contrast; similarly for *C* and *D*, *C* being slightly concave upward. Since *E* is on the same side of the bar as *A* and *C* we might compare them. It is not very far from being a composite of *A* and *C*, although the left end is rather low.

Comparing *F* with *B* and *D* we find that the convexity upward of *B*, which is more conspicuous in *D*, has nearly disappeared in *F*, the maximum point having fallen well below the dip average.

A of Plate XII is a fair composite of *E* and *F* on Plate XI, and lies between them on the bar. It is nearly a straight line, the variation being a slight concavity upward. *B* on Plate XII is the composite for bar 53, and, although it shows wide range between individual assays, indicates a bar more nearly uniform than any of the preceding. The curve is slightly concave upward although the average value of drillings is very close to that of the dips.

In the diagrams on Plates X, XI and XII, the curves are bent toward the horizontal at the ends, but this feature is inconspicuous in the final product.

If a curve were constructed according to the fourth method suggested at the beginning (warping all portions of the curve between the maxima and minima toward the dip average) the result would be rather striking and extremely improbable. This would be especially conspicuous for bars 49, 50 and 51.

For these reasons two diagrams are given for composite values of bars 52 and 53, constructed according to the first and fifth methods, the zigzag line of short straight segments being used rather than the smooth curve. The composite diagrams for both of these bars show average points which are less extreme, measured from the dip average, than those for bars 49, 50 and 51, and for these reasons were chosen. The intermediate and extreme points are chosen so that the areas above the dip average but below the profile are equal to the areas below the dip average but above the profile. Or considering the areas between the dip average and the profile as positive or negative according as they are above or below the former, the positive areas equal the negative. Some such balance must exist if the dip average represents the true assay value of the bar. Such a diagram does not show, necessarily, the exact location or manner of such variations, but it does give a minimum variation. Possibly much of this balance may be accomplished in the outer inch of the bar, rather than within this profile. Complete adjustment in this outer portion, however, seems improbable to the writer, especially in profiles such as Plates II, *B*, IV, *B*, and VI, *B*. In profiles such as Plates IX, *D*, and XII, *B*, where values in the profile are both below and above the dip average, it seems much more reasonable to suppose that an approximate balance is maintained within the profile. The same may be true for Plates II, *B*,

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IV, *B*, and VI, *B*, in which cases the local variations are extreme in places. Although the gradation is almost surely not along straight lines, the turning points may be as sharp as those in the diagram, and are not necessarily curved crests or troughs.

In both profiles on Plate XIII one end of the diagram shows a rising value, whereas the other end shows a decreasing value. Neither extreme is far from the dip average but the erratic variation is thus shown. These two bars, however, are less heterogeneous, or variable through a smaller range, than many cyanide bars, and although an average of all the drillings is close to that of the dips in these two cases, it is decidedly otherwise in many other instances, such as bars 49, 50 and 51, as well as many of the bars listed between 1 and 48.

Another interesting feature brought out by these diagrams is the variation in assay value of individual samples. In the simple diagrams this variation seems equally conspicuous along the profile (or bar) and does not appear to favor any single portion of the bar. The composite diagrams cannot of course be here considered, as there is superposition of points, due to projection on the medial plane.

In a study of these curves a few characteristics seem to be common, but no attempt will be made to offer other than tentative reasons for such. It is to be hoped, however, that a further detailed examination may be undertaken to clear up these points. Such an examination is of necessity confined to the government mints (or possibly assay offices) but for some reason the men in these places have been handicapped by a lack, either of time or of a desire for such an investigation. Considerable extra time is required for such detailed sampling, and still more elaborate methods must be pursued before a satisfactory solution is reached. This is out of the question in the assay offices as the time element is an important feature, and also on account of the bookkeeping system used, no sampling being possible after settlement on a bar is made. At the mints an unusual opportunity is afforded, and such work would be of considerable value to the service as well as of scientific interest.

Summary

1. Drillings taken near an edge are extremely variable, but generally assay higher than the dips, a fact which may be due to early solidification and an accompanying segregation in those portions of the bar of metals of the highest freezing points, taking into account mutual solubilities and alloying tendencies. In a platinum-gold alloy this same characteristic is very apparent, but is due, as Mathey observes, to the liquation of the platinum.

2. The center of the bar, top and bottom, is usually lower in value than the dips. As suggested early in this paper, this may be due to a late solidification in these and similar points, which results in a higher per-

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centage of base metals with low freezing points. Mutual solubilities may be a factor here, also, but the substance in large excess (gold) does not concentrate toward the center or last place of solidification as does quartz in the cooling of a quartz-rich magma.

3. In a majority of cases the top drillings assay higher than do those of the bottom, and the same is true of the dips. In order to balance the personal equation the method of weighing up assays was reversed so that the weigher who received the top samples at first, later received the bottom samples. For this reason samples were frequently exchanged in weighing. The same variations, however, continued to be apparent.

An extreme range in the dip assays is in some cases due to the presence of impurities introduced while the dips are being taken. Where a scorifier is used to catch part of the stream of metal, small pieces of clay might easily be caught and retained in the bullion granules.

4. Drillings taken at any one point along these profiles vary as widely in assay value as drillings of any other point similarly chosen. Also, maxima and minima points are as apt to occur near the center of the bar as near the ends. From this standpoint the corner drillings seem no better nor worse than drillings from nearer the center.

5. The liquation in some cyanide bars is similar to that which takes place when the rare metals are present in appreciable amounts, but in the former instance it seems to be due to the presence of certain base metals.

Attention is again called to the fact that these difficulties are not met with where a small amount of silver is present, as the presence of this latter metal seems to diminish segregation. Lessened segregation is probably due to the solvent properties of silver toward certain base metals such as lead, zinc, etc. Silver may exert an appreciable influence on other metals or non-metals also. Where 90 or 100 points of silver were known to be present in the bar, no dip samples were taken, and the drillings rarely failed to check well within the limits allowed. This is doubtless due to the fact that silver or silver-gold alloy in which the silver content is nearly 10 per cent., or better, is a good solvent for the metals or non-metals causing the segregation. Pure or silver-free gold is not a good solvent for these substances and hence does not prevent the formation of an irregular bar.

This suggests one way to overcome the difficulty of assaying cyanide bullion, although it does not solve the problem. If to all such cyanide bars known to contain little or no silver, a small amount of the metal be added in melting, a uniform bar after melt could be easily obtained and its fineness accurately determined. From the fineness thus obtained the inquartation silver could be subtracted, leaving the silver fineness of the original bar. As such gold bars are alloyed with silver for refining operations, the above suggested operation would mean a little additional work for the computers only, but this extra work would be more than offset by the increased accuracy of the gold assay.

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Discussion

Frederic P. Dewey, Washington, D. C. (communication to the Secretary*).—I wish to congratulate Professor Hance and to thank him for his extremely valuable addition to the fund of data upon this subject. It presents clearly conclusive and incontrovertible proof of the unreliability of drill samples of this class of metal for determining the value of such bullion. I am afraid, however, that his hope that some one in the mint service might continue such an investigation upon a more elaborate and exhaustive scale is, for the present at least, doomed to disappointment.

As a practical matter, the mint service is not now particularly interested in segregation in crude bullion. So much trouble and friction arose, both in the original purchase and in the transfer from one institution to another, in handling segregated bullion, that it became necessary to adopt drastic measures and now it is seldom attempted to determine the value of such bullion in the service. When bullion known or even suspected of being segregated is presented for purchase, it is at once strongly refined in the pot and the disturbing elements so far removed that concordant assays may certainly be obtained from the dip samples and a reasonably fair agreement obtained from drill samples. The mint service prefers that the owner should refine his bullion himself, but if he does not we will do it at his expense and return the slag to him when desired.

Much work has been done in the mint service trying to arrive at the value of deposits of segregated bullion without strong refining. It was no unusual thing for such deposits to be melted three times at the purchasing office and to have 30 to 40 assays on the metal made before buying it. Yet when shipped to a mint it was found necessary to melt two or three times more and to make 15 to 25 assays before the mint would accept the bar, and there were considerable differences between the valuations of the two offices. In what was undoubtedly the worst case ever investigated, the purchasing office melted the deposit, 953 oz., three times, with a loss of 105.5 oz., and made 46 assays; the mint melted three times, with a loss of 6.6 oz., and made 28 assays. The mint allowed the purchasing office 1.178 oz. of fine gold more than the purchasing office claimed, while an extensive investigation of the mint samples by the Bureau, in which over 100 independent assays were made in various service laboratories, indicated that the metal carried 0.7 to 0.9 oz. more of fine gold than the mint allowed. Preliminary assays on the original metal indicated that it was about 661 fine in gold. Even after all the assays made on the mint samples, I would not be absolutely positive as to the composition of the final bar, but it may be taken as close to 751 fine in

* Received Feb. 7, 1916.

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gold, 60 in silver and 189 in base. Undoubtedly more refining was required in order to obtain thoroughly reliable and satisfactory assays. Subsequently a bar from the same mill was refined to over 900 fine in gold by the purchasing office and accepted at the mint, on six assays, at the purchasing office valuation.

One office expended a tremendous amount of energy upon this question. For six months it was attempted to determine the gold in all such deposits, several hundred of them, as received, for the purpose of comparison with the amount paid for after melting and more or less treatment by the office. There is a sprinkling of cases where the difference was within reason, $10 either way, but a very large proportion of them showed a much wider variation. Some of them were startling and these were not by any means all on one side. In nearly 100 cases, selected at random and tabulated for an entirely different purpose, there were five cases of plus differences and six cases of minus differences of over $200 each. The highest plus variation was $609.43. The highest minus variation was $413.76. Owing to the many and serious difficulties of determining the value of the metal as received, it would not be at all proper to speak of these differences as gains and losses. The best that can be done in this line is to call them apparent gains and losses, but even this is not always entirely warranted.

In some cases the metal was received in small bars and several melted together, but this by no means secured agreement in the two valuations. Six bars from 79.78 to 157.01 oz. in weight were melted together, with a loss of 10.63 oz., but the assays varied from 349.9 to 352.6 fine in gold, and the bar was remelted, with a further loss of 43.13 oz. when the gold assays showed from 371.3 to 371.9 fine, upon which the bar was reported at 371.5 fine in gold, the silver being 557 fine and the base 71.5. Four assays had been made on each one of the small bars, making a total of 24, but there was an apparent loss of $201.51 shown by the final bar. The maximum plus difference noted above was on a mass melt of three bars, about 200 oz. each, totalling 609.83 oz., which was melted with a loss of 73.57 oz. The refined bar showed 908 fine in gold, 62 in silver and 30 base.

Twelve small bars weighing 39.70 to 50.15 oz. each, and totalling 548.38 oz. were melted together and held under blast for 4.5 hr. with a loss of 21.93 oz. in weight and an apparent loss of $67.54, but the gold assays varied from 917.9 to 919.9 fine. The bar was remelted, with a further loss of 7.17 oz., but with an apparent gain of $8.87 in value over the first melting. There was an apparent loss of $58.67 on the calculated value of the 12 individual bars. The final bar showed 932 fine in gold, 48 silver, and 20 base.

Four bars ranging from 156.88 to 223.07 oz. and totalling 751.10 oz. were each assayed four times showing wide variations in the sets of gold assays. Roughly speaking, the silver could be said to average approxi-

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mately 48 fine and the base 130. Upon being melted together, with a loss of 37.29 oz. the gold assays ranged from 863.2 to 865.7 fine. The apparent loss in value was $7.69. On a second melting with a loss of 22.66 oz. the gold assays varied from 892.4 to 894.4, and there was an apparent gain of $2.42 on the figured value of the four individual bars. On a third melting with a further loss of 21.21 oz. the gold assays ranged from 917.6 to 918 fine, and there was a final apparent loss of $47.92 on the estimated value of the four bars. The final bar was 55 fine in silver and 27.5 base.

After this investigation the office abandoned all attempts to nurse along such metal by mild treatment in stages and proceeded at once to strongly refine all suspicious bars. In general, the method consists in blowing air onto the surface of the molten metal, adding an acid flux as necessary, thickening occasionally with bone ash and skimming.

Recently the shippers have supplied their assays of 14 bars for comparison. Curiously the assay office results showed gains of gold over the shipper's figures in seven cases and losses in seven cases. As received, these bars varied in weight from 800 to 1,100 oz. with a total of 13,122.72 oz. After refining they weighed 736 to 1,043 oz. totalling 12,125.83. The total loss in melting was 996.89 oz. or an average of 71.2 oz. per bar. The individual loss in melting ranged from 50.75 to 89.40 oz. per bar. As received they varied from 240.5 to 301.1 fine in gold, 556.3 to 628.9 in silver and 122.5 to 173.5 in base. After refining, the apparent gold gains ranged from 1.519 to 5.601 oz. on a bar, the total apparent gain being 22.128 oz. The apparent gold losses varied from 0.932 to 4.485 oz. on a bar and totalled 15.659 oz. The net apparent gain of gold was 6.467 oz. In the case of the silver there were 12 apparent gains ranging from 1.44 to 11.45 oz. on a melt and totalling 64.58 oz. The two apparent silver losses were 3.33 and 23.78 oz. The net apparent gain of silver was 37.47 oz. The slags were returned to the depositor and undoubtedly he realized further gains from them. These figures appear to show a substantial gain to the shipper by having his bullion strongly refined before being purchased by our assay office, but they emphasize most emphatically the difficulty of correctly determining the amount of gold present in the original unrefined bullion.

A great many of our troublesome bars have been over 100 fine in silver and some of them very much over. Eight bars ranging from 43.42 to 138.04 oz. and totalling 702.30 oz. were each assayed for gold four times. Naturally, the range of the assays on the smallest bar was not so very much, being only from 275.4 to 276.1, but on the largest bar it varied from 282 to 288.4. The silver estimates ranged from 570 to 620 and the base from 84.5 to 144. On melting these bars, with a loss of 52.89 oz., the gold assays ranged from 306.3 to 309.3. It is scarcely worth while to compare the value of this bar with the computed values of the

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individual bars. On remelting the bar, with a loss of 28.5 oz. the gold assays ranged from 322.1 to 323.1, with an apparent gain of $8.19 in value. On a second melting, with a further loss of 3.5 oz. the gold assays ranged from 324.3 to 325, with a further apparent gain of $2.26. The final bar was 661 fine in silver and 14.5 in base.

Nine other bars from the same mill totalled 670.63 oz. The mass melt weighed 644.51 and the gold assays ranged from 277.6 to 282.3 fine. On remelting, with a loss of 52.22 oz. the gold assays ranged from 300 to 301.4 fine. On a second melting, with a further loss of 15.93 oz., the gold assays ranged from 308.4 to 309 fine. The final bar was 672 fine in silver and 19.5 in base.

There is, of course, no comparison between this metal and that so successfully treated with silver by Professor Hance, but seven small bars totalling 270.47 oz. yielded a mass melt weighing 250.40 oz. on which the gold assays varied from 742.5 to 745.6 fine. On remelting, with a further loss of 14 oz. the gold assays varied from 786.8 to 789.5 fine. On a second remelt, with a further loss of 17.35 oz. the gold assays varied from 846.9 to 847.8 fine. The final bar showed 100 fine in silver and 53.5 base. A bar weighing 726.93 yielded four gold assays varying from 807.4 to 811.1, the silver being approximately 115 and the base 76. On melting, with a loss of 42.37 oz. the four gold assays varied from 855.7 to 855.9, the silver fineness was 120 and the base 24.5.

It is therefore apparent that much more investigation is required in order to determine the class or classes of gold bullion in which segregation can be overcome or reduced by the addition of silver.

GEORGE C. STONE, New York, N. Y.—I do not know whether the gold people have tried the method used by the zinc men in sampling. We have found that drilling was very unsatisfactory. Ordinary spelter segregates pretty badly; the lead is irregularly distributed, and there is always more at the bottom of the slab, so that when we drill we drill completely through, but we have found that the much more satisfactory way with spelter was to saw the slab, and by taking saw cuts from each side more than half way across, we obtained samples that would check with each other, and give a most excellent material for the laboratory. The sawdust is so fine that it is easily mixed, and several duplicate samples can be weighed out and checked with each other perfectly, and samples taken by different people will check extremely well.

I have never heard of it being used in connection with gold bullion, but I think it would be worth experimenting with. Some of the men here have had experience with the two methods of sampling.

FRANCIS P. SINN, Palmerton, Pa.—A number of years ago we were using the method of sampling spelter which Mr. Stone spoke of, drilling the slabs. The chief trouble that we had was keeping the drills in shape

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and in cutting up the strips made by the drills. Our present method of using an ordinary band-saw has saved us a lot of time not only in taking the sample from the slab but the sample requires practically no preparation. It has saved us both time and money. I should say that where we are making 100 tons of spelter a day it saves us the labor of three or four men every day.

GEORGE C. STONE:—Mr. Sinn did not mention that he is making the high-grade spelter, of which the analysis has to be within very close limits, and they (New Jersey Zinc Co. of Pa.) take more samples, probably, for the tonnage than at any other works.

E. G. SPILSBURY.—What number of samples are supposed to be taken of this high-grade spelter and what percentage of the slabs are taken as samples?

FRANCIS P. SINN.—In shipping our spelter, we take one slab from every 20 that go in a car, and make three cuts in that slab. We take a cut half through on one side at one end, half through on the same side at the other end, and half through the middle from the other side, so we feel that we cover the slab pretty well in that way, and do not destroy the slab itself by allowing it to break. We are sampling our spelter also as it is made, that is, we sample it hot. We use a small dipper, take a small sample from each ladle, and drop this into water, and the spelter breaks up into a small sort of spatter which is not exactly granulated. In this way we get a sample from every bit of spelter that is made, a little being taken from every ladle that is poured.

MR. SMITH.—I would like to ask Mr. Sinn how he takes the sample from the ladle?

FRANCIS P. SINN.—We use a small iron dipper and are criticized, of course, by the casual observer. He would say that a certain amount of iron would get into the sample from the dipper. As a matter of fact, a skull forms on the inside of the dipper that protects the sample. We always form that skull in the dipper before keeping the sample. It is really a galvanized dipper.

MR. SMITH.—Is the sample taken from the bottom of the ladle or from the spelter, as poured?

FRANCIS P. SINN.—Our experience has been that there is no possibility of actually settling the lead in spelter that runs under 1 per cent. of lead, and we are making spelter running considerably under 1 per cent. When we started our method of sampling, we took separate samples from each part of the ladle as it was poured and thoroughly tested out the conditions, but we take the samples from the top of the

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ladle now. If the spelter runs over 1 per cent. of lead there is a possibility of not getting an accurate sample in this way.

H. L. GLENN, Seattle, Wash. (communication to the Secretary*).—I quite agree with the author that in determining the values of bullion the dip samples are better than drills. In order to avoid the possible errors to which he refers, caused by particles of the scorifier-dipper getting into the sample, I suggest the use of a device which we have found comparatively satisfactory in the United States Assay Office in Seattle: Instead of using a scorifier to take the dip samples we use a graphite stirring rod. In the side of this rod near the lower end is bored out a small pocket which will hold an ounce or more of the molten bullion. With this stirring-rod dipper we are able to take the two dip samples, one from near the top and the other from near the bottom, without delay and with no danger of the difficulties which the author experienced.

Regarding the method of arriving at the top and bottom values of the bar: I prefer to assay each sample separately rather than to mix the samples as suggested by the author. Then, after assaying, if it is desired, an average of the two results may be taken, which secures the same result but still keeps the samples as originally taken, for future use.

The Seattle Assay Office purchases only a small amount of cyanide bullion annually and what we do purchase has, invariably, a larger proportion of silver than is found in the Salt Lake bullion. We have, however, the same trouble as was found by Professor Hance, though the discrepancies in assays are not as great, which is probably accounted for by the larger proportion of silver in the bullion. The Mint Bureau is under obligations to Professor Hance for his painstaking and satisfactory work.

I regret that time will not permit of a more extended discussion of this paper. On the whole the author has handled his subject most admirably and I hope before long that some one may have the time and patience to make further experiments in order to solve some of the problems which Professor Hance was obliged to drop for lack of time.

JAMES H. HANCE, Iowa City, Ia. (communication to the Secretary†).—Sampling gold-bullion bars according to the same method found so successful with zinc bullion or spelter would doubtless yield much more nearly uniform assay values than drilling methods. For several pertinent reasons I have never tried this method, but I believe it would assist in getting true values, if *solid bars only* are available. The dip samples, however, are more easily obtained where melting or remelting is done, and, except in *rare* instances, are sufficiently accurate if carefully prepared and properly used.

* Received Feb. 21, 1916.
† Received Mar. 13, 1916.

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In many cases cyanide bars are extremely brittle, and a bar with three slots, each half way through the bar, might be too fragile for ordinary handling. Packing and shipment from assay offices to the mints would also increase the liability of breakage.

Another difficulty, not necessarily serious, is manipulation of such sampling. Since all weights are recorded to hundredths of a troy ounce, all fragments must be saved, and this would be more difficult with saw samples than with drillings. A more serious phase of this would be probable salting or contamination of the sample if the saw were not perfectly clean each time it was used. Cleaning the saw teeth carefully each time would be absolutely necessary, and would take time. The time factor, although not of primary importance, would also be appreciably greater for this method of sampling, and where the bars are numerous and the help limited, this time element might be an important consideration. Finally, I am doubtful if this method would yield results of greater accuracy, if as great, than those of dip sampling.

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A Development of Practical Substitutes for Platinum and its Alloys, with Special Reference to Alloys of Tungsten and Molybdenum*

BY FRANK ALFRED FAHRENWALD, CLEVELAND, OHIO

(New York Meeting, February, 1916)

I. INTRODUCTORY

METALLURGICAL research has discovered many an alloy possessing properties not combined in any single metal, and progress still consists chiefly in the investigation and utilization of alloys. In the case of iron, the demands of automobiles, high-speed machines and high-duty engines have led to the production of special iron alloys which will meet any reasonable specifications in that field. In like manner, the bronzes, brasses and other alloys of copper have been brought to remarkable perfection, and for nearly every industrial purpose some alloy has been found more suitable than the pure metal.

Less complete success has attended the attempt to find substitutes for gold, platinum, and the other precious metals. Indeed, it is not likely that a material can be produced which will possess all the properties of any one of them; yet it is reasonable to hope that, for any given use, the properties required may be found in some less expensive material. Thus, in incandescent electric lamps, the wire passing through the thick glass neck of the bulb was, until recently, almost universally made of platinum, for the single reason that no other known material, suitable as a conductor, had the same coefficient of expansion as glass. But a comparatively recent investigation of the iron-nickel series[1] has shown that alloys of those metals may be produced, the coefficient of expansion of which can be accurately controlled between that of iron or nickel and zero. Thus, in that particular industry, a substitute for platinum has been found.

According to statistical reports, not only considerable quantities of gold and iridium, but also more than one-third of the annual supply of platinum, are used (and, in the nature of the case, irrevocably lost) by dentists. Platinum is thus employed in several forms. As a thin foil, it serves various purposes for which its high melting point, pliability, chemical resistance, and other properties—including the ease with which it may be soldered—are invaluable. But it is most extensively used in

* Condensed from a dissertation submitted in partial fulfillment of the requirements for the degree of Doctor of Philosophy in the University of Michigan.

[1] C. E. Guillaume: Non-expansive Alloys, *The Metallographist*, vol. vi, p. 162, 1903. Grenet and Charpy: Dilatation of Steels at High Temperatures, *The Metallographist*, vol. vi, p. 238, 1903.

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the alloy with iridium, which is more resistant chemically than pure platinum, solders as readily, and possesses, besides, the quality of stiffness, and is not seriously softened by annealing at ordinary soldering temperatures. These advantages dictate its use, in spite of its high cost. The discovery of a substitute in dentistry for platinum and platinum-iridium, especially if it possessed useful properties which they lack, would be eagerly welcomed, and would find wide application in other arts also. The solution of this problem has been undertaken by the Research Foundation of the National Dental Association, for which the work described in this paper was done by the writer.†

The substitute desired must satisfy the following conditions:

1. Its melting point must be high, at least well above 1,200°C.

2. It must not be affected by those chemical compounds formed in its application, nor should it oxidize at a soldering temperature.

3. It must possess sufficient strength to resist stresses tending to change its form while in place, and at the same time be sufficiently pliable to be worked to the desired shape.

4. Its coefficient of expansion must be low, in order that desired dimensions may be easily produced in the finished product. (This factor is important, since the range through which this material is manipulated is often more than 1,000°C.)

5. It should unite readily with gold, silver and similar metals, and their solders.

6. Its cost of production should be low, as compared with that of platinum.

After the entire list of metals had been considered with respect to these conditions, it was evident that any search for the desired material must be among the alloys, since experience has shown that the physical properties of a metal may be radically changed by the addition of varying amounts of another element, or of several elements, as in the case of steels, brasses and bronzes.

Considerations based upon the periodic law of atomic weights, and the table constructed in accordance therewith by Mendeléeff and Lothar Meyer, led to the conclusion that the field for profitable research was narrowed to the elements chrome, manganese, iron, cobalt, nickel, copper, silver, palladium, gold, molybdenum and tungsten, and their alloys (ruthenium ridium and osmium, likewise theoretically indicated, being ignored because of their costliness and scarcity).[2]

† This work was done in the Department of Mining and Metallurgy, Case School of Applied Science, and the Department of Chemical Engineering, University of Michigan.

[2] [NOTE.—The original thesis contains an interesting discussion of the theoretical considerations above mentioned. It assumes the probability that the physical properties (such as the melting points) as well as the chemical (such as base-forming or acid-forming) properties of the elements bear certain relations to their arrangement according to the periodic law. This discussion has been omitted here to save space.

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When two or more metals are brought together in the liquid state, they mix exactly like two ordinary liquids. When the temperature is lowered the solidified mass may contain any one of the four following constituents: Pure components; solid solutions; compounds; and eutectics—or some combination of these.

A comparison of the properties of different alloys containing these constituents has shown that they impart their characteristic properties to the alloy of which they form a part; in fact, the relation between the constitution of an alloy and its mechanical properties is so clearly defined that the possibilities of industrial application may be predicted for a given alloy, if its constituents are definitely known. Conversely, if a certain application is desired, as in the problem under consideration, a definite limit may be placed upon the number and amount of constituents permissible.

Fortunately, the number of constituents is limited to four, as given above. Pure metals impart their own characteristics; solid solutions are, in general, the ductile constituents (if formed of ductile metals or of a preponderance of one ductile metal); compounds and (usually) eutectics are hard and brittle, while the latter, even when present in very small amounts, tend to solidify between the grains of the alloy, thus destroying its ductility.

The problem was to determine what combination, if any, of the elements named, in whatever form, would meet the assumed set of specifications.

Accordingly, each of the 11 elements was combined, in varying proportions, with each of the other 10, giving 55 binary series to be considered with regard to their suitability as practical substitutes for platinum and platinum-iridium alloys.

II. Work on Alloys Made by Fusion

Behavior under the hammer, or when drawn through draw plates, and under the influence of acids and alkalies, was sufficient to indicate whether any particular specimen should be discarded at once or made the subject of further investigation.

It was not considered necessary to measure the various melting temperatures encountered, as only a complete fusion, free from all contaminants, was desired. The purity of all components was the highest obtainable, while the purity of the resulting alloys was checked by microscopical and, if necessary, by chemical analysis.

1. *Apparatus*

The Gran-Annular electric furnace[3] was used in this set of experiments, and served as no other type of furnace would have done under

[3] Zay Jeffries: Notes on the Gran-Annular Electrical Furnace, *Metallurgical and Chemical Engineering*, vol. xii, p. 154 (1914); also, C. H. Fulton: *Trans.*, vol. xliv, p. 769 (1912).

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conditions involving constant use, quick heating and cooling, sensitive control, and low upkeep cost. This is of the granular-carbon resistance type, in which temperatures are limited only by the melting point of the

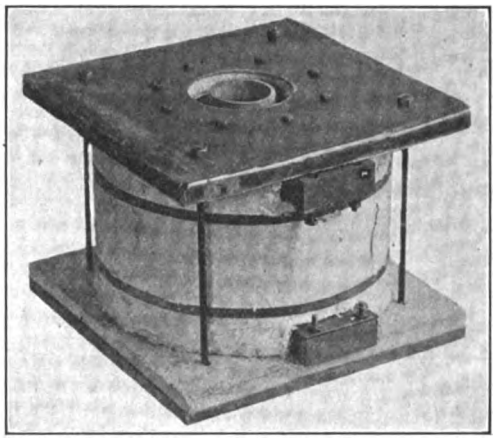

FIG. 1.—THE GRAN-ANNULAR ELECTRIC FURNACE.

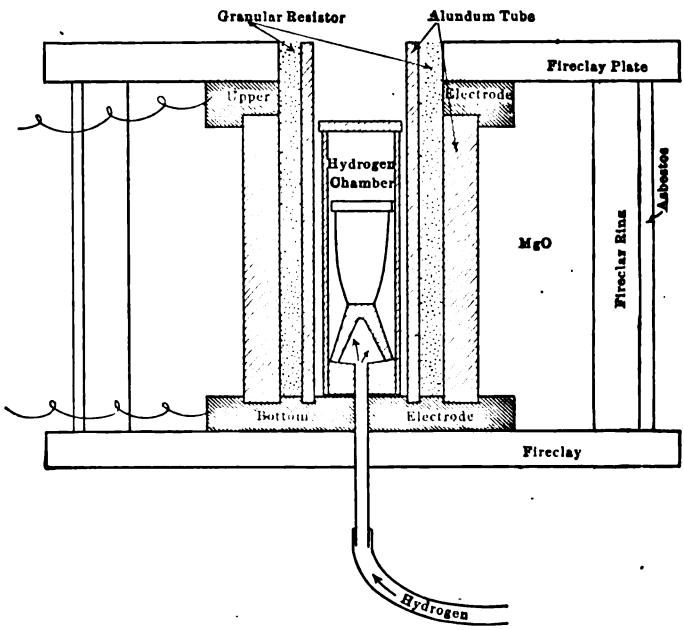

FIG. 2.—VERTICAL SECTION OF FIG 1.

tubes used. With tubes of alundum, it can be safely run up to 1,800°C. This furnace is shown in Figs. 1 and 2, the latter a vertical section, showing also the means adopted for introducing any desired atmosphere into the crucible chamber. This latter device consists of a cylindrical alun-

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dum crucible, fitted with a Marquard tube which passes through the bottom of the furnace and thence to the generator, provided with close-fitting cover, so that a gas may be maintained within it, and surrounding the crucible containing the melt, at a pressure sufficient to cause an outward flow through all the pores of the chamber walls, thus effectually preventing contamination by the atmosphere of the outer furnace chamber.

When this furnace is run above 1,200°C., the atmosphere within the heating chamber is of the following analysis: O, 0.20 per cent.; N, 68.90; CO_2, 0.70; CO, 30.20 per cent.

TABLE I.—*Materials, Etc., Employed in Experiments*

Metal	Crucible Material	Protective Cover	Inert Atmosphere
Chromium	Carbonaceous material not permissible.		H_2 or N_2.
Molybdenum...	Magnesia may be used.		
Tungsten.....	Alundum questionable.		
Manganese.....	Magnesia, or a lining of magnesia. Alundum questionable. Silica or carbon must not be present.	$BaCl_2$.	H_2 or N_2.
Iron.....	Porcelain, magnesia, graphite with magnesia lining, alundum. Carbon or silica should not be in contact with fusion.	$BaCl_2$. Gypsum.	H_2. N_2 forms nitride. CO or CO_2 will carbonize.
Cobalt.....	Same as for iron, except that porcelain is strongly attacked by oxide formed.	Same as for iron.	H_2 or N_2.
Copper.....	Charcoal, graphite, porcelain, fire-clay, magnesia, silica, alundum.	KCl plus NaCl, borax, KON, charcoal.	H_2, N_2, CO. CO_2, illuminating gas.
Silver.....	Nickel, iron, fire-clay, graphite, porcelain, alundum.	KCl plus NaCl, borax, KCu, charcoal.	Same as for copper.
Gold.....	Practically any non-metallic crucible.	None necessary.	None necessary.
Palladium	Magnesia, alundum. Carbonaceous material should not be present.	Glass.	H_2 forms alloy with palladium, although at lower tempertaure.

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The tests for the general properties of the alloys, such as malleability, ductility, corrodibility, etc., are so simple as to require ordinary laboratory appliances only.

Table I gives a list of crucible materials, protective atmospheres, and covers, which have been found satisfactory in fusing these metals.

2. *Results*

Some of the 55 binary series, as, for instance, those high in iron, manganese, or chromium, indicated at once their inability to meet specifications, while others, as those of nickel-palladium, nickel-tungsten, etc.

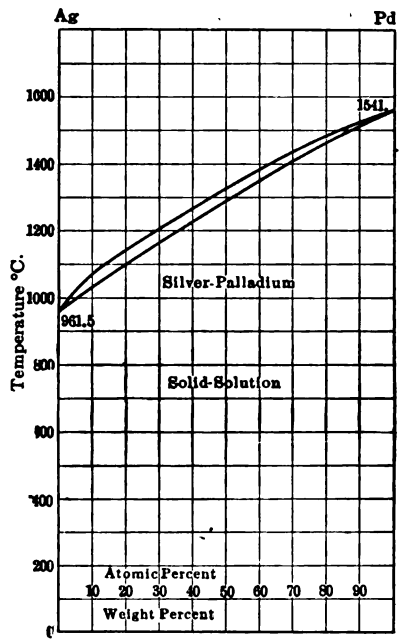

FIG. 3.—EQUILIBRIUM DIAGRAM OF THE SILVER-PALLADIUM SERIES.

required careful tests, and many degrees of concentration, to determine their comparative properties.

The investigation of these series (with the exception of those high in tungsten or molybdenum, which are not adaptable to fusion methods) is represented by nearly a thousand melts, which are unequally distributed among the different series, the number of fusions necessary in each series depending upon how nearly it approached the standard specifications.

The numerous negative results are not reported here, because they would greatly increase the volume, without correspondingly adding to the value, of this paper. It is sufficient to say that of these 55 series, only those of palladium with gold and silver have proved practically valuable.

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The equilibrium diagram, Fig. 3, of the silver-palladium series[4] shows a complete series of solid solutions. Alloys of these, in any proportions, have been found to be very soft, chemically inert, and if chosen of a composition to give a sufficiently high melting point, meet the necessary requirements of a foil to replace platinum, in the field of dentistry. The cost of this material will range from about 1 to 50 per cent. of that of platinum, depending upon the melting point desired.

The palladium-gold alloys (Fig. 4), however, are superior to those of silver-palladium, as gold is superior to silver. As in the palladium-silver series, the range between liquid and solid is very narrow, resulting in little

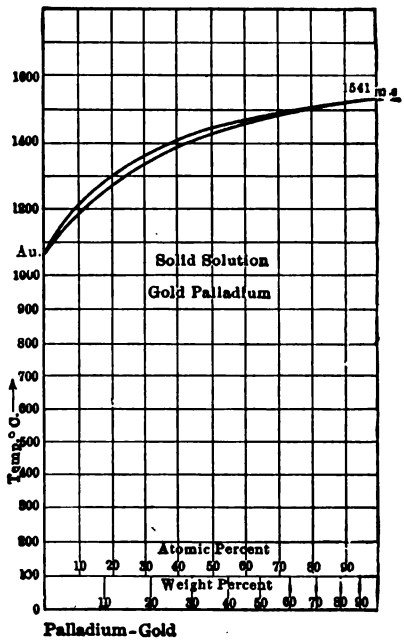

FIG. 4.—EQUILIBRIUM DIAGRAM OF THE GOLD-PALLADIUM SERIES.

tendency toward segregation. Fig. 5 shows curves having melting points as ordinates and cost per cubic centimeter, of silver-palladium and gold-palladium alloys, as abscissæ.

Metals are usually price-listed by weight; consequently an erroneous idea is generally prevalent as to the actual relative cost of different available materials, when applied to a given operation. A certain vessel, for instance, will require a definite volume of material, irrespective of its weight, which depends upon its density. Suppose, for example, that the volume of this vessel be 1 c.c. To fill this with platinum will cost, at

[4] Reur: Ueber die Legierungen des Palladiums mit Silber, *Zeitschrift für Anorganische Chemie*, vol. li, p. 315 (1906).

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present quotations, about $30.50; with gold, $12.50; and with palladium, about $22. This is not the popular conception of the relative cost of these elements which, when compared by weight, is gold, $20.67; platinum, $48; and palladium, about $60 per ounce.

In connection with this series is a phenomenon for which no explanation is evident; 1 per cent. of palladium darkens gold perceptibly; at from 2 to 3 per cent., the color is a dull light bronze; while at about 10 per cent., no trace of a yellow color can be detected.

In the case of the silver-palladium alloys, a yellow tinge is perceptible

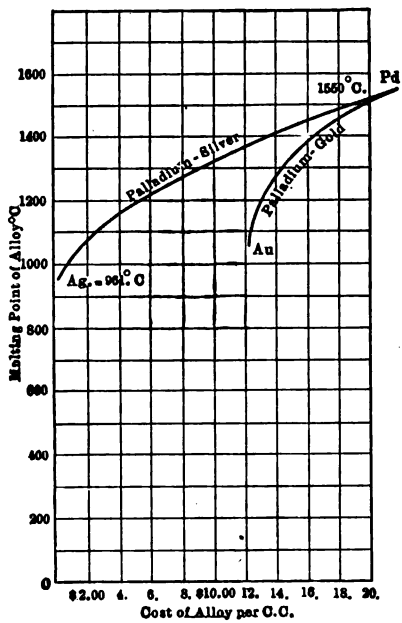

FIG. 5.—CURVES SHOWING RELATIVE COST OF PALLADIUM-SILVER AND PALLADIUM-GOLD ALLOYS HAVING EQUAL MELTING POINTS.

throughout the series, which phenomenon is equally hard to explain, considering the pure whiteness of both components.

Although platinum, in its softer forms, as in foil, may be replaced by the above-described palladium alloys, investigation to this point had produced nothing in the nature of a hard, strong, non-oxidizable, non-corrodible material which would serve as a substitute for the platinum-iridium alloys.

III. TUNGSTEN AND MOLYBDENUM

The investigation thus far had considered, both singly and in binary combination, the entire list of available metals with the exception of tungsten and molybdenum, the treatment of which, by ordinary fusion methods, was not feasible.

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It may be said in advance that any treatment described in this chapter as applying to tungsten will apply likewise to molybdenum. Also, an outline of the properties of tungsten will include those of molybdenum, which will not be given in detail. Molybdenum is not so hard, not so stiff, and not so chemically resistant as tungsten; but its properties parallel those of tungsten so closely as to permit the omission of their detailed description.

1. *Properties of Tungsten and Molybdenum*

Tungsten is popularly best known in its application in the manufacture of incandescent lamps, and more technically, in the form of its alloys with iron and other metals forming special steels. Its alloys with cobalt, chromium, and similar metals, forming alloys of the "Stellite" type,[5] have also found application.

The remarkable properties of the pure, metallic, ductile tungsten are, however, continually enlarging its field of application. This material is practically insoluble[6] in any of the common acids; its melting point is higher than that of any other metal, its tensile strength exceeds that of steel; it is para-magnetic; it can be drawn to smaller sizes than any other metal, and its specific gravity is 70 per cent. greater than that of lead.

Wrought tungsten has been substituted with success for platinum and platinum-iridium, as contact points in spark coils, voltage regulators, telegraph relays, and for similar purposes. By reason of its greater hardness, higher heat conductivity, and lower vapor pressure, it gives much longer service than platinum. In the automobile industry, for instance, platinum has been in many cases entirely replaced for this purpose by tungsten.

Tungsten gauze is used successfully for filtering acid liquors and where fumes are encountered,[7] while acid-proof dishes and tubes are also made of tungsten.[8]

Its tensile strength varies from 300,000 to 650,000 lb. per square inch; its thermal conductivity is more than twice that of platinum; its hardness varies from 4.5 to 8.0 (Mohr's scale), and its thermal coefficient of expansion is only 4.3 by 10^{-6}. That for molybdenum is yet

[5] Described by C. L. Sargent: *Journal of the American Chemical Society*, vol. xxii, p. 783 (1900); and by E. Haynes: *Iron Trade Review*, vol. li, p. 927 (1912).

[6] W. E. Ruder: Solubility of Wrought Tungsten and Molybdenum, *Journal of the American Chemical Society*, vol. xxxiv, p. 387 (1912).

[7] C. Ehrenfeld: A Study of the Chemical Behavior of Tungsten and Molybdenum and Their Oxides, *Journal of the American Chemical Society*, vol. xvii, p. 381 (1895).

[8] C. G. Fink: *Transactions of American Electrochemical Society*, vol. xvii, p. 229 (1910); and the *Proceedings of Eighth International Congress of Applied Chemistry*, vol. xxvi, p. 503 (1912). Also, I. Langmuir: *Transactions of American Electrochemical Society*, vol. xx, p. 237 (1911); and W D Coolidge: *Transactions of the American Institute of Electrical Engineers*, vol. xxix, Part ii, p. 961 (1910).

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lower, 3.6 by 10^{-6}, while for platinum this value is 8 by 10^{-6}. For many other metals this ranges above 13 to 14 by 10^{-6}.

In short, ductile tungsten meets all but two of the preliminary set of requirements. The exceptions are, that it oxidizes easily at a red heat, and that it does not solder with gold and its alloys, except under strongly reducing conditions. Moreover, it was found that the larger-sized wires of this material were quite brittle, and even those sizes suitable for the purpose specified contained treacherous spots.

But the field of possibilities had, at this stage, narrowed itself to a consideration of this metal, with molybdenum as a second choice. It was therefore necessary to find: First, some means of preventing its oxidation; second, some means of increasing its affinity for gold, silver and their alloys; and third, some method of producing it in more reliable form.

2. *Coating with Precious Metals*

In earlier work on the gold and palladium series, it was discovered in attempting to alloy tungsten with these metals that while the larger pieces of tungsten were not appreciably dissolved, they were nevertheless removed from the molten bath with a beautiful impervious coating. As either of these furnished the surface qualities which tungsten lacked, the two remaining conditions seemed to be fulfilled. And so they were; but after this bath the tungsten wires were as brittle as glass.

TABLE II.—*Experiments in Coating Tungsten with Gold, Silver, and Palladium, under Varying Conditions*

Composition			Time	Temperature of Bath							
Per Cent.			Seconds	1,100°C.		1,350°C.		1,500°C.		1,600°C.	
Ag	Au	Pd		Effect on Tungsten	Coating	Effect on Tungsten	Coating	Effect on Tungsten	Coating	Effect on Tungsten	Coating
100	.....	.....	10	None...	Very poor.	None...	Very poor.	Very brittle.	Poor.	Very brittle.	Poor.
100	.....	.....	30	None...	Very poor.	Quite brittle.	Very poor.	Very brittle.	Poor.	Very brittle.	Poor.
	100	.....	10	None...	Good.	None...	Good.	Slightly brittle.	Good.	Very brittle.	Good.
	100	.....	30	None...	Good.	None...	Good.	Quite brittle	Good.	Very brittle.	Good.
		100	10	.....	.....	.....	.....	.....	.....	Very brittle.	Good.
		25	10	.....	.....	Quite brittle.	Good.	Very brittle.	Good.	Very brittle.	Good.
		50	10	.....	.....	.....	.....	Very brittle.	Good.	Very brittle.	Good.
	50	50	10	.....	.....	.....	.....	Very brittle.	Good.	Very brittle.	Good.
	75	25	10	.....	.....	.....	.....	Very brittle.	Good.	Very brittle.	Good.
	90	10	10		.....	None...	Good.	Very brittle.	Good.	Very brittle.	Good.
	95	5	10	.....	.....	None...	Good.	Very brittle.	Good.	Very brittle.	.....

These tests were made on drawn wires of 30-mil size, which without the coating, and heated in an atmosphere of hydrogen, crystallised and became brittle at 1,700°C. in from 10 to 20 min.

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The problem thus arose of applying this coating, of such material, and under such conditions, that the other valuable properties of the tungsten would not be destroyed during the process. The crystallization of the tungsten was obviously due to one of three, or a combination of three causes; improper coating temperature, improper composition of the coating material, or improper time element.

In order to determine whether brittleness was due to the metal of the bath, or to the factor of time or temperature, tests were made, using varying composition and temperature of bath, and allowing the material to remain in the bath for varying periods.

Typical results, taken from the notes on these experiments, are given in Table II.

As shown in Table II, pure gold gives the best results. Both silver and palladium seem to accelerate crystallization, for some reason not

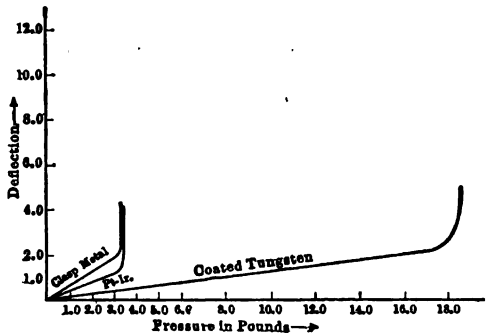

FIG. 6.—DIAGRAM SHOWING COMPARATIVE STIFFNESS OF COATED TUNGSTEN, "CLASP METAL," AND PLATINUM-IRIDIUM ALLOY CONTAINING 30 PER CENT. IRIDIUM.

apparent at the present time. Silver has practically no affinity for tungsten, under ordinary conditions; gold will take up small amounts with difficulty; while palladium will readily dissolve certain amounts of it.

Pure gold, alone, forms a beautiful adherent coating which would serve in ordinary cases; but by adding small amounts of palladium to the coating bath, its melting point is raised, and, at the same time, a better bond between the tungsten and this protective layer is obtained.

Most of this coating was performed in an open atmosphere. The tungsten was first dipped into a bath of fused sodium carbonate, which operation caused a cleansing and protective shell of this material to cling to the specimen when it was withdrawn. This flux was automatically removed when the tungsten was dropped into the metal bath.

Properties of the Coated Material.—The material produced in this manner presents the surface properties of pure gold, or its alloys, together with the original internal properties of pure tungsten, a combination

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which cannot be approached by any other metal or alloy at present available. With few exceptions, the alloys of platinum and iridium are inferior to this coated tungsten.

This is because of the great strength and stiffness of this new material, and due to the fact that it can be soldered, or otherwise manipulated, at high temperatures, without softening or losing this stiffness and elasticity to the slightest degree.

A comparison of the stiffness of coated tungsten with that of similar test pieces, one of which is an alloy containing 70 per cent. of Pt, and 30 per cent. of Ir, and the other the so-called "clasp-metal," an alloy of gold, copper and platinum, is shown in Fig. 6. In making these experiments the test wires were supported upon two knife edges 0.5 in. apart, weight being applied upon a third, midway between these two.

The chief remaining objection to this material is its frequent brittleness and unreliability in the larger-sized (above 40 to 50 mils) wires. The quality of this material, as supplied, however, is constantly improving. A discussion of the probable causes of this brittleness, and a suggested method for improvement are given below.

IV. Alloys of Tungsten and Molybdenum

1. *The Production of Ductile Tungsten and Molybdenum.*

The extreme brittleness of tungsten and molybdenum, when produced from the molten state, seems also to be an unavoidable characteristic of their high percentage alloys, when produced in a similar manner. However, in view of the remarkable success which has been attained in the production of these metals in the ductile form, by using methods not involving a preliminary molten condition, and because of the remarkable properties of these metals when so produced, it seemed advisable to determine whether the few undesirable properties could not be properly modified by introducing varying small amounts of other elements, using methods similar to those employed in the production of pure metal.

It was thought that perhaps the addition of small amounts of molybdenum would decrease the extreme brittleness of tungsten, and perhaps result in a material more pliable and ductile in the larger masses, or that the presence of small amounts of the more noble metals would lessen its tendency to oxidize; perhaps even prevent oxidization below relatively high temperatures, and at the same time add such surface qualities that it could be readily brazed or soldered with gold and other precious metals, under ordinary atmospheric conditions.

A description of the manufacture of ductile tungsten, as taken from the literature and patent specifications, gives the operation essentially as follows: The pure tungstic oxide, in certain cases containing a small percentage of ThO_2 (the effect of which will be discussed later), is reduced

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in an atmosphere of hydrogen. This reduced powder is compressed into briquets, about 0.5 by 0.5 by 15 cm. in size, which are first sintered at about 1,300°C. and then heated electrically to a temperature near the melting point of the tungsten; after which, by successive swagings at temperatures above a red heat, the material is compacted and welded to a solid metallic mass, and when reduced to about 30 to 40 mils in diameter, it is drawn through diamond or ruby dies, first hot, and finally cold. All operations involving temperatures above a very dull red are performed in an atmosphere of hydrogen, or nitrogen.

No reference has been found describing any sort of metallographic control in connection with these various operations, and so far as available information indicates, the above method for the production of ductile tungsten and molybdenum has been developed and employed without assistance from this branch of science. This will, no doubt, account for the difficulty which has apparently been encountered in adapting it to the production of the material of larger sizes in ductile form, and in eliminating the last traces of brittleness in the drawn wires.

No manufacturer of ductile tungsten has been able, as yet, to supply our laboratory with specimens of ductile high-percentage alloys of tungsten. Directly because of this fact, and with confidence in the final success of a method involving proper metallographic control during the various stages of manufacture, experiments were begun to determine the conditions under which such alloys might be produced.

2. *The Internal Structure of Metals*[9]

Pure metals, in the solid state, are aggregates of crystals. These do not as a rule possess any regular geometrical shape, but the individual crystalline grains forming this aggregate possess the essential character

[9] Sir J. Alfred Ewing: The Inner Structure of Simple Metals, *Journal of the Institute of Metals*, viii, p. 4 (1912).

Ewing and Rosenhain: The Crystalline Structure of Metals, *Philosophical Transactions*, vol. cxciii, p. 353 (1900); vol. ci, p. 279 (1901).

J. E. Stead: The Crystalline Structure of Iron and Steel, *Journal of the Iron and Steel Institute*, vol. i, p. 145 (1898).

See also the following for relevant matter:

R. Hooke: *Micrographic*, London, 1665.

Reaumer: *L'Art der convertu le fer forge en acier*, Paris, 1722.

Sorby: *British Association Report*, 1864.

Osmond and Werth: Des Propriétés de l'Acier, *Annales des Mines*, Series 8, vol. viii, p. 6 (1885).

M. F. Osmond: Sur la Cristallographie du Fer, *Annales des Mines*, Series 9, vol. xvii, p. 110 (1900).

James Thomson: *Philosophical Trans.*, Glasgow (1882).

Behrens: *Das Mikroscopische Gefüge der Metalle und Legierungen.*

Lehmann: *Molekularphysik*, vol. i, p. 279.

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of all crystals, in having a regular arrangement, or orientation, of matter within their boundaries.

Fig. 7 is a photograph of the cast surface of pure gold, which has been slowly and carefully cooled, showing how the interior orientation of molecules has influenced the configuration of the surface of each grain. This mass of metal is seen to be made up of two or three crystal groups.

In Fig. 8, the surface is seen to be composed of a larger number of separate grains, irregular both in size and shape. This is a photograph of a more quickly cooled specimen. In this the grains are distinguished not only by these irregular boundaries but by a difference in texture between one grain and another; some are bright, some are dark, while others range

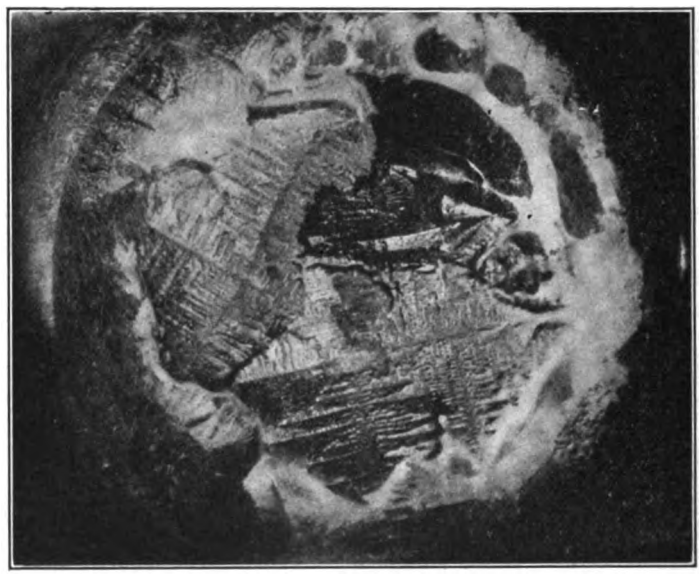

FIG. 7.—CAST SURFACE OF PURE GOLD SLOWLY COOLED. × 6 DIAMETERS.

intermediately. If the source of light used in making this micrograph be moved, the brightness of the individual grains will change, as shown in Fig. 9; those which appeared bright or dark under the first condition of lighting will assume a different shade when illuminated from a different angle. This change is due to the fact that the surface exposes a multitude of little facets, or planes, all facing one way in any one grain, but of a different inclination in the different grains. The brightness of each grain depends upon the relative angles of these reflector systems.

Surface conditions indicate that the entire volume of any one grain consists of an assemblage of structural units, which may be likened to the bricks of a wall, all parallel in any one grain but facing differently in the different grains. This has been further verified by a study of cross-sec-

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tions of metals, and by observation of the fact that certain crystals have been found to act as a three-dimension diffraction grating toward x-rays.

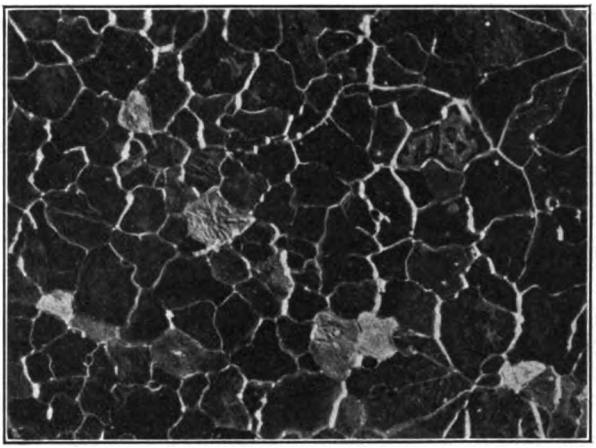

FIG. 8.—CAST SURFACE OF GOLD MORE QUICKLY COOLED. × 40 DIAMETERS.

Each grain, then, is a crystal, imperfectly formed, it is true, because it has grown simultaneously from separate nuclei, symmetrical growth being stopped by final contact with neighboring crystals, after which only such

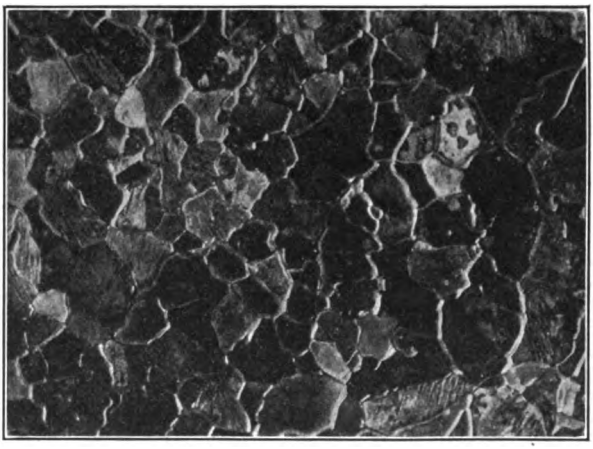

FIG. 9.—SAME AS FIG. 8, UNDER LIGHT AT A DIFFERENT ANGLE.

parts of the crystal have grown as would serve to fill up the liquid spaces remaining.

Other explanations[10] have been advanced for the formation of crystal-

[10] C. H. Desch: Solidification of Metals from the Liquid State, *Journal of the Institute of Metals*, vol. xi, p. 57 (1914). (This contains complete references.)

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line grains, but the assumption that growth proceeds simultaneously from separate nuclei receives the strongest support.

These true crystal grains are found also in metals which have been shaped by cold working. Fig. 10, showing a longitudinal section of swaged tungsten wire, illustrates this rather poorly, but it is given here to show the cold-worked structure of this material. The grains are seen to have been elongated.

Many experiments have been performed by many scientists, in attempting to determine what phenomena take place during cold working, by virtue of which the crystal grain would permit such distortion without the destruction of its general character. The studies of Rosenhain and Ewing,[11] who made a microscopic examination of polished metallic pieces during the actual straining operation, seem most satisfactorily to

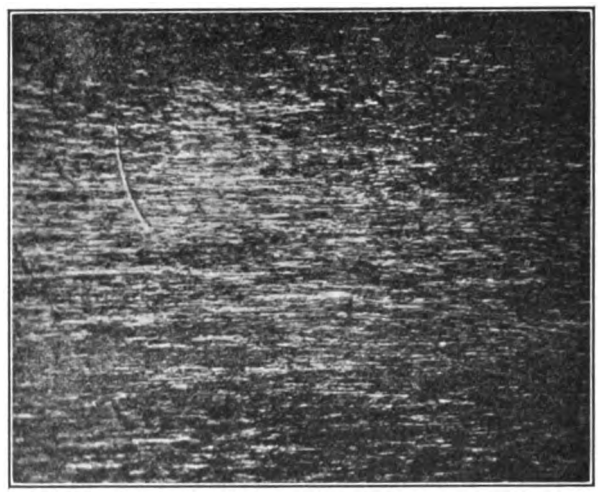

FIG. 10.—LONGITUDINAL SECTION OF SWAGED TUNGSTEN WIRE. × 770.

reveal the internal changes which take place. It was noticed during these tests that a number of fine dark lines appeared across the surface of each grain, being parallel in each but running in different directions in the different grains. This is illustrated in Fig. 11 which shows the surface of a distorted bead of cast gold. It was found that these lines or bands were, in reality, small steps produced by shearing at a corresponding number of internal surfaces. It was shown that each crystal grain behaves as does a pack of cards when bent or otherwise distorted, *i.e.*, by the sliding of very thin layers upon each other, so producing the "stepping" at the edges. These observed lines were called slip bands, and by their formation the grains are capable of a certain amount of deformation without breaking up.

[11] Ewing and Rosenhain: *Philosophical Transactions*, vol. cxciii, p. 355 (1900); and *Proceedings of the Royal Society*, vol. lxv, p. 85 (1899).

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To explain how this distortion may probably take place within the mass of the crystal itself which, as a crystal, will not permit of flow, several theories[12] have been advanced, which, when combined and correlated, seem to clear up the matter very satisfactorily, for working purposes at least. It is well known that unequal pressure, when applied to a solid phase alone, lowers its melting point. It should be possible, then, by sufficiently increasing this pressure, to cause the material to melt at ordinary temperatures. Since both liquid and solid phases would at this point be in equilibrium, the molecular activity, or molecular freedom, of each phase would be the same. By equating values representing these molecular activities,[13] and integrating, with the substitution of proper thermodynamic values, an equation has been evolved by Johnston and

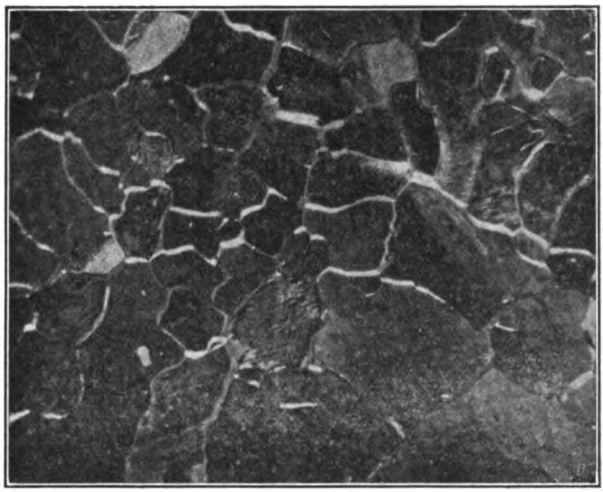

FIG. 11.—SURFACE OF A DISTORTED BEAD OF CAST GOLD. × 77.

Adams, involving the density, heat of fusion, and normal melting temperature, from which may be calculated the pressures necessary to cause the different metals to melt at ordinary temperatures. This calculation resolves itself into the following final equation:

$$P = 95.1 \times Q \times D \times \text{Log} \frac{T_1}{T}$$

[12] G. T. Beilby: The Hard and Soft States in Metals, *Journal of the Institute of Metals*, vol. vi, p. 5 (1911); see also *Proceedings of the Royal Society*, vol. lxxix, p. 463 (1907), *Proceedings of the Faraday Society*, June, 1904, and *Philosophical Magazine*, vol. viii, p. 258 (1905). Johnston and Adams: High Pressure on the Physical and Chemical Behavior of Solids, *American Journal of Science*, vol. xxxv, p. 209 (1913).

[13] Lewis: A New System of Thermodynamic Chemistry, *Zeitschrift für Physische Chemie*, vol. lxi, p. 129 (1907); also, *Proceedings of the American Academy of Science*, vol. xliii, p. 259 (1907).

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in which P is the pressure, in atmospheres, necessary to cause fusion; Q is the heat of fusion of the compressed material; D, its density; T_1, its normal melting temperature (Kelvin); and T, the temperature at which it is desired to cause the material to melt.

Johnston and Adams give the value of P for several metals, as follows: Sn, 2,200; Bi, 3,000; Cd, 3,300; Al, 5,100; Zn, 6,900; Ag, 14,000; Cu, 24,000; Pd, 31,000; Pt, 46,000 atmospheres.

The fact that layers of a solid, many hundreds of molecules in thickness, can really have this mobility of the liquid state conferred upon them by purely mechanical movement was first proved by Beilby,[14] who found that a true skin is formed over the surface of a metal during polishing. This skin was found to be distinctly different from the crystalline material beneath it; it was harder, and even when formed on the surface of a crystal, of which the hardness varied in different directions, its hardness was the same in all directions. It was also found to be more readily attacked by solvents.

In polishing, and in any cold-working operation, it is conceivable that those particles which bear the brunt of the strain will be under a pressure corresponding to that at which they will melt at ordinary temperatures, and so will give way, flowing into areas of lesser pressure where resolidification takes place, leaving the load to be taken up by others in succession. According to this theory, during deformation of a metal, amorphous material is formed at all internal surfaces of shear (as indicated by the formation of slip bands), serving as a cementing material which is both harder and stronger than the crystalline material which it surrounds. This effect is illustrated in the increased tensile strength and stiffness of drawn wires.

Ordinarily this distortion is not carried to the stage at which the crystals are completely converted to the amorphous form, and hence the mass consists of one component in two phases, one of which is in a metastable condition. In the case of ordinarily undercooled liquids, the presence of the stable form tends to produce true equilibrium immediately, but in the case of the metals, the sluggishness of the molecules at this low temperature prevents this transformation. However, if the temperature be raised, the kinetic energy of the molecules rises, until, at a certain point, the fragments of crystals which remain are able to impress their orientation upon the molecules of the amorphous substance, and so slowly "absorb" it.

It will be evident that, above this temperature, the kinetic energy of the molecules is so great as to permit their returning at once to the crystalline form when displaced by mechanical distortion. The manipu-

[14] G. T. Beilby: Surface Flow in Solids, *British Association Report*, Glasgow (1901); and The Surface Structure of Solids, *Journal of the Society of Chemical Industry*, vol. xxii, p. 1166 (1903).

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lation of metals below this temperature is the familiar "cold-working" operation; above this point, "hot-working." The crystalline state is that of maximum thermal stability and of minimum mechanical stability, while for the amorphous material the reverse is true.

According to the "amorphous cement" theory, it should follow, then, that a metal which has been cold-drawn to that stage which represents the formation of the maximum amount of amorphous material possible will be in the best condition to meet the imposed requirements of elasticity and stiffness.

Intercrystalline Boundaries.—The discussion has thus far included only such changes as may take place within the mass of the crystal itself, without a consideration of the phenomena by virtue of which the separate grains, or crystallites, are held together.

The internal strength of a single crystal, within which the molecules are arranged in a manner involving a minimum of potential energy, and hence of intra-molecular distance, may be ascribed to cohesion due to a mutual attraction of the closely packed molecules. Where two crystals meet, however, it seems improbable that the molecules of the different systems of orientation are close enough to one another to permit a degree of mutual attraction comparable to that existing between molecules of the same crystal. It might be supposed that the intercrystalline boundaries would be surfaces across which cohesion acted less strongly than it does within the mass of the crystal.

These intercrystalline boundaries would then be regarded as planes of weakness, when compared to the strength across any plane within the crystal. But this is not true, for the fracture of metals, in normal condition, always runs across the crystal grain[15] rather than around the boundaries; and it is almost universally conceded, by those familiar with the microstructure of metals, that a fine-grained structure (in which these boundaries are numerous) is superior in reliability and strength to a coarse-grained one—which indicates that the intercrystalline boundaries are planes, not of weakness, but of strength. This has been ascribed by some to an interlocking of the irregular crystallites themselves, and by others to other causes; but a theory[16] which receives strong support supposes the existence of a cementing material[17] between the crystal grains. This theory assumes that when a metal crystallizes from the liquid state, many layers of molecules, lying at the planes where growing

[15] W. Rosenhain: Deformation and Fracture in Iron and Steel, *Journal Iron and Steel Institute*, vol. lxx, Part II, p. 212 (1906).

[16] G. D. Bengough: A Study of the Properties of Alloys at High Temperatures, *Journal Institute of Metals*, vol. vii, p. 9 (1912), and Discussion by Rosenhain, p. 176; also J. A Sears: On the Longitudinal Impact of Metal Bars with Rounded Ends, *Transactions Cambridge Philosophical Society*, vol. xxi, p. 105 (1908).

[17] Rosenhain and Ewen: Intercrystalline Cohesion in Metals, *Journal of the Institute of Metals*, vol. viii, p. 194 (1912) and vol. x, p. 119 (1913).

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crystals meet, do not crystallize, perhaps because of the mutually neutralizing effect of adjacent crystal systems, and so solidify in the vitreous amorphous form. That this view is not untenable is shown in the case of amorphous silica, silicates, and similar substances[18] which occur normally in the crystalline state also. The Phase Rule has, however, developed an excessive tendency to regard the stability which results from the crystalline state as the chief if not the only kind of stability which may exist under ordinary conditions in substances *normally* crystalline.

Such materials as glass, vitreous silica, etc., are harder and usually more brittle, but also much stronger, than the same substances in the crystalline form. Although these substances are essentially of the nature of liquids, they do not possess the mobility ordinarily associated with that state. Their viscosity at ordinary temperatures is very great, but they do possess the power of flowing to some extent, as shown by the bending of glass tubing under its own weight when placed at an angle for some time; and this property has been shown to exist even in vitreous silica.[19]

Substances of this type solidify to a completely amorphous mass because of the extreme sluggishness of their molecules at and below the freezing temperature, so that cooling may be comparatively slow without resultant formation of crystals.

In the case of metals, however, because of their extreme molten fluidity and long crystallization range below the freezing point, it has been found impossible to cool them from the molten state so quickly as to produce a completely amorphous solid. This has made it difficult to obtain direct experimental verification of the "amorphous cement" theory, as advanced in explanation of the strength of intercrystalline boundaries and the increased strength and hardness of cold-worked metals. It has been admittedly impossible, with the means at present available, to answer this question by any attempt to produce completely amorphous metals from the molten or crystalline states. However, by adopting a different method of attack, comparable to that employed in synthetic processes, the writer has, in a manner, overcome this difficulty—sufficiently so, at least, to point out the remarkable way in which the physical properties of a simple metal may vary with its internal structure.

New Experimental Evidence in Support of the "Amorphous Cement" Theory (the Influence of Phase upon the Physical Properties of Gold)

According to the "amorphous cement" theory, then, the strength of the intercrystalline boundaries in metals is due to the presence of

[18] Tamman: *Krystallisiren und Schmelzen.*

[19] Kay: Thermal Hysteresis of Fused Silica, *Philosophical Magazine*, Series 6, vol. xx, p. 718 (October, 1910).

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material which has not crystallized, and the increased strength of cold-worked metals is due to the formation of the amorphous phase during manipulation.

If the assumption that greater strength in metals is due to amorphous material present be sound, then a metallic mass known to contain the amorphous phase should possess properties differing from those of the metal when in the crystalline state.

The scheme adopted for testing this assumption involved the preparation of a series of gold briquets, made up of gold particles representing a different degree of subdivision for each briquet of the series, from coarsely crystalline to as near molecular proportions as could be obtained. This series consisted of four groups, each comprising several specimens for the different tests. No. 1 was made of pure cast gold, and represented a purely crystalline condition. No. 2 was made of fine gold filings, which represented a certain amount of amorphous material produced by extreme cold-working conditions. No. 3 was prepared from the precipitate formed by ferrous sulphate, and represented a very fine state of division.

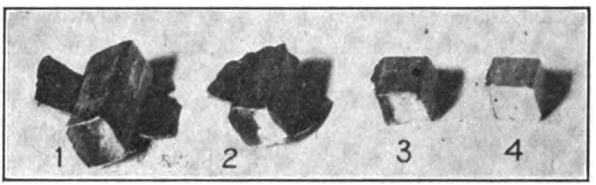

FIG. 12.—FLOW UNDER PRESSURE OF GOLD BRIQUETS OF DIFFERENT STRUCTURE.

No. 4 represented the ultimate limit of practicable subdivision. These briquets were prepared from a mud of colloidal gold, produced as follows: A deep red colloidal gold solution was prepared by reduction with 0.5 per cent. solution of hydroxylamine hydrochloride. This solution was carefully dialyzed to remove alkali salts, then evaporated to a "mud," and further dried on a water bath. It was found difficult to prepare a red solution containing more than 0.05 per cent. of gold, and even this, upon concentration, became greenish in color and resulted finally in a grayish-green sludge.

The size of gold particle, as determined for a red-gold colloidal solution, is of about 10 to 20 micro-microns. This size was no doubt increased, as indicated by a change in color, during evaporation. About 7 liters of this dilute solution was reduced for each briquet.

The apparatus used for briqueting this material was the same as that used for tungsten and molybdenum, and is described below.

The behavior of each of these series is strikingly shown in the photographs, Figs. 12 and 13. The different extent to which each would flow through the small openings between the different parts of the briquet

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press is remarkable. Nos. 1, 2 and 3 were made under the same pressure, 9,700 kg. per square centimeter (about 135,000 lb. per square inch), and contain the same weight of gold. In the case of No. 4, the utmost pressure available, 19,300 kg. per square centimeter (about 265,000 lb. per square inch) did not cause any indication of flow. A similar striking difference is shown in the case of the Brinell hardness number of each of these series, using the large ball and a pressure of 500 kg. No. 1 was too soft to bear this weight, so the hardness number was taken from a bar of rolled and annealed gold and found to be 23.80. The hardness number for No. 2 was 38.10; for No. 3, 53.40; and for No. 4, 94.70. These depressions are shown comparatively in Fig. 13.

Other striking phenomena were noted in connection with these series, but their behavior under pressure will serve to point out the fact that much remains to be discovered regarding the physical properties of simple metals, as well as what may be expected should this knowledge lead to a control of phase.

The important feature of these results, however, is the apparent support they give to the amorphous cement theory of Beilby, Rosenhain, and others, above. Since these results have shown that characteristics

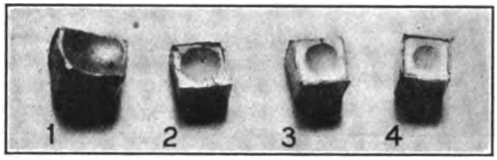

FIG. 13.—COMPARATIVE HARDNESS OF BRIQUETS IN FIG. 12.

of an amorphous metal are great hardness and resistance to deformation, nothing could more nearly explain the increased strength of the cold-worked metals than a theory assuming the formation of amorphous material during manipulation.

3. *Practical Deductions*

With the application of these crystallographic considerations, the probable cause for the difficulties encountered in the production of ductile tungsten, and to a greater extent, of ductile tungsten alloys, seemed apparent. Careful microscopical examination of many specimens of drawn wire submitted to this laboratory revealed in all cases that brittleness is accompanied by a difference in crystal structure. Fig. 10, already described, showed a longitudinal section of drawn tungsten wire, which was very tough and flexible. Fig. 14, a longitudinal section of a similar wire, which was exceedingly brittle, clearly shows the discontinuity of the drawn, fiber-like structure.

In discussing the internal structure of metals it was pointed out that a sufficiently high temperature would permit the complete crystallization

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of amorphous metal. The larger crystals of this mass may impress their orientation upon the molecules of smaller crystals, thus causing crystal growth; and mechanical distortion, above a certain temperature, will not greatly reduce this grain size.

In devising a method for producing tungsten and its alloys in a non-brittle form, these points were regarded as fundamental. The brittleness of the examined specimens of drawn tungsten was regarded as due to an insufficient breaking up of its crystalline structure. Hence, in producing this material, or its alloys, in ductile form, it would be necessary to eliminate excessive crystalline material; and in order to accomplish this, the following means suggested themselves, based upon deductions drawn from earlier conclusions regarding the behavior of worked metals in general.

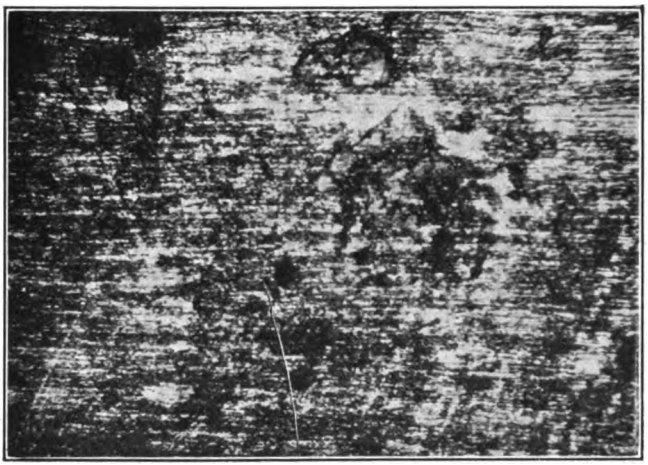

FIG. 14.—LONGITUDINAL SECTION OF BRITTLE TUNGSTEN WIRE. × 743.

1. In order to produce a practical minimum of crystal size in the starting ingot, the treating temperature must be as low as possible.

2. In order to decrease the amount of crystalline material, cold-working conditions must prevail.

3. In order to introduce sufficient distortion, with its accompanying formation of the maximum amount of amorphous material, resulting in greater ductility in the larger sized wires, drawing should be commenced at an earlier stage in the reduction of the briquet, because this operation is much more effective in breaking up a crystalline structure than is swaging.

4. For obvious reasons, the above operation should be controlled by microscopical examination. Internal structure, not temperature, is the treating criterion.

That these premises are fundamentally sound is shown by the following results of experimental work.

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V. Experiments

The experimental work, which involved the application of the above considerations, and the results of which are outlined in this paper, included investigations of the following series: pure tungsten, tungsten containing 0.75 per cent. ThO_2, pure molybdenum, tungsten-gold, tungsten-palladium, tungsten-molybdenum.

The method adopted comprised the compression of amorphous powder to briquet form; the heat treatment of the compressed material; and the forging of the resulting ingot.

The tungsten and molybdenum used in these experiments were obtained through the courtesy of the Cleveland branch of the General Electric Co., and were the chemically pure material used for the production of drawn wires.

The tungsten was of two batches, one containing 0.75 per cent. of ThO_2 which is the material largely used for lamp filaments, while the

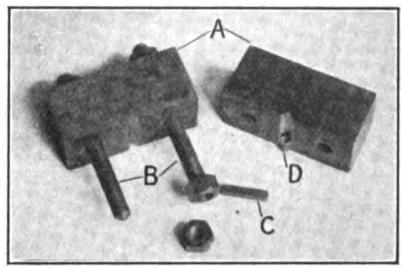

Fig. 15.—Briquet Mold.

other was tungsten in which no trace of impurities could be detected. The gold and palladium were purified in this laboratory by several re-precipitations. The preparation of the amorphous material, from which briquets were made, will be described in connection with the different series.

1. *Compressing Powder to Briquet Form.*

The mold used in producing the raw briquet is shown in Fig. 15. This consists of two halves of hardened tool steel, held together by strong bolts. Into each half is milled a polished V-shaped groove, so that when the two parts are bolted together a square chamber is produced. Into one end of this chamber the plug *D* is inserted, and after filling the chamber with the material to be compressed, the plunger *C*, is forced into place. This plunger is steel containing tungsten, 16.5; chromium, 2; and carbon, 0.6 per cent., which, after heat treatment, has been tested under pressures exceeding 300,000 lb. per square inch. The face of the plunger was highly polished so that one surface of the raw briquet was available for micro-examination.

The design shown in Fig. 16 was a failure, since the adhesion between the briquet and the semi-cylindrical surface of the mold was greater

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than the cohesion across the section within the briquet itself, causing it to split while being removed.

Compression was obtained in a Brinell hardness-testing machine, the

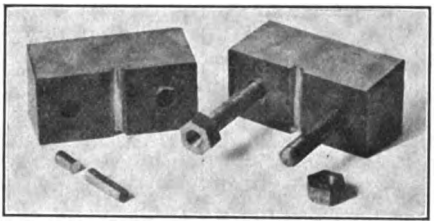

FIG. 16.—UNSATISFACTORY BRIQUET MOLD.

ball, with its socket, being replaced by a special head. The press *A* was placed upon the adjustable table as shown in Fig. 17, so that the plunger came axially under the piston of the machine. This machine gave very

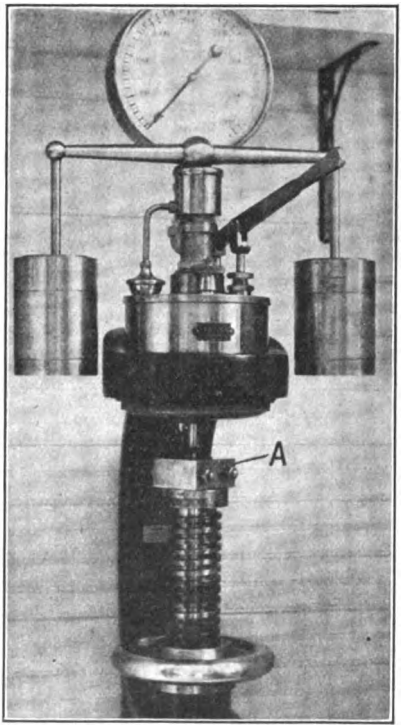

FIG. 17.—MACHINE FOR COMPRESSION OF BRIQUETS.

sensitive and accurate control up to a total load of 4,000 kg. Various total loads, when exerted upon the piston of 0.207024 sq. cm. in cross-section, are given in comparative values as follows:

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Total Pressure, Kilograms	Kilograms per Square Centimeter	Atmospheres	Pounds per Square Inch
500	2,412.7	2,335.5	33,561
1,000	4,825.5	4,671.0	67,122
2,000	9,651.0	9,342.0	134,244
3,000	14,476.5	14,013.0	201,366
3,750	18,095.5	17,516.0	251,707
4,000	19,302.0	18,684.0	268,488

2. *Heat Treatment of the Compressed Material*

Quartz-Tube Furnace.—Fig. 18 shows a quartz-tube furnace which was used for the reduction of tungsten and molybdenum, and other oxides

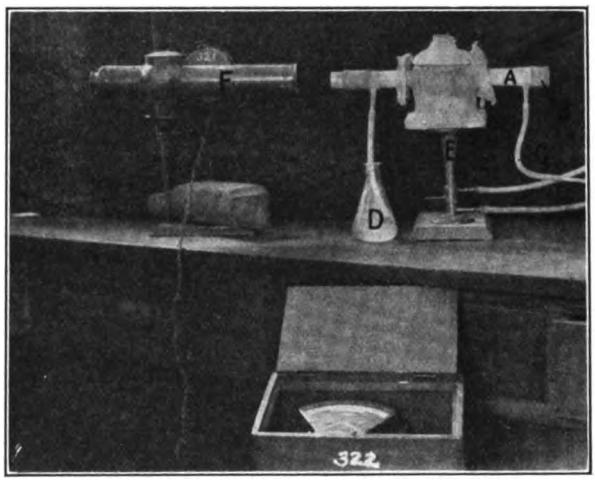

FIG. 18.—QUARTZ-TUBE FURNACE.

and salts; also for various purposes where heating in controlled atmosphere was desired but where very high temperatures were not necessary. In this, temperatures up to 1,225°C. are easily and quickly reached. The fused quartz tube, *A*, has a glass window at one end, and at the feed end the plug, *B*, which can be removed for the insertion of the boat containing the material to be treated. Hydrogen, or other gas, enters through tube *C* and escapes through the bubbler *D*. A Meker blast lamp, *E*, was used for heating. The method of sighting with optical pyrometer *F* is also shown.

Gran-Annular Electric Furnace.—For temperatures up to 1,800°C. this furnace far surpasses any other type which the writer has used in experimental work of this kind.

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Figs. 1 and 2 showed this furnace in detail. Fig. 19 shows a setup of the furnace, *A*, in series with cast-grid rheostat, ammeter, and switchboard. An optical pyrometer, *B*, points directly into the heating chamber of the furnace.

Electrical Resistance Furnace (Resistance of Charge).—The above furnace equipment, however, did not provide for the treatment of tungsten and molybdenum briquets, or those containing high percentages of these metals. Moreover, it provided no means for forging these ingots. This operation had to be carried out at high temperatures and in an inert atmosphere. Fig. 20 gives a vertical section of a furnace so designed as to

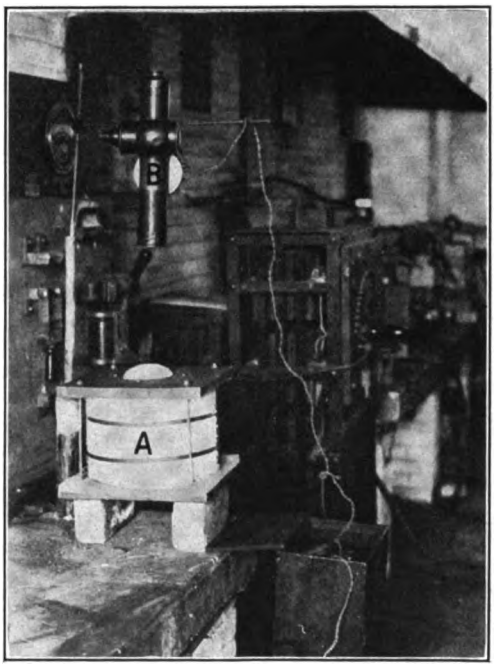

FIG. 19.—GRAN-ANNULAR ELECTRIC FURNACE WITH CONNECTED APPARATUS.

furnish temperatures ranging from that of the furnace room up to, and above 3,000°C., depending upon the material of the briquet, and also to serve as a forge for working the material into a solid mass while at a high temperature. This consists essentially of two adjustable electrode bars, *A* and *B*, fitted with fused tungsten contact surfaces, *C*, between which the briquet, *D*, is placed. Current is passed through the briquet which, by its own resistance and that between the contact surfaces, may be heated to any temperature up to its melting point. An inclosed heating chamber, *E*, is formed by dropping the sliding cup, *F*, into a close-fitting recess made for it at the base. Gas is passed through the tube, *G*, into this chamber,

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from which it escapes through the outlet shown. This opening serves also as a peep hole for the optical pyrometer.

In operating, the bar, *A*, with its electrode and housing, *F*, is raised through its guides, *H* and *I*. The briquet, *D*, is then placed in position on the lower electrode, *C*, after which the upper electrode is lowered to contact. The inverted cup, *F*, is then lowered into place, forming the chamber *E*, through which is passed an inert gas, swiftly at first, to expel the air, and

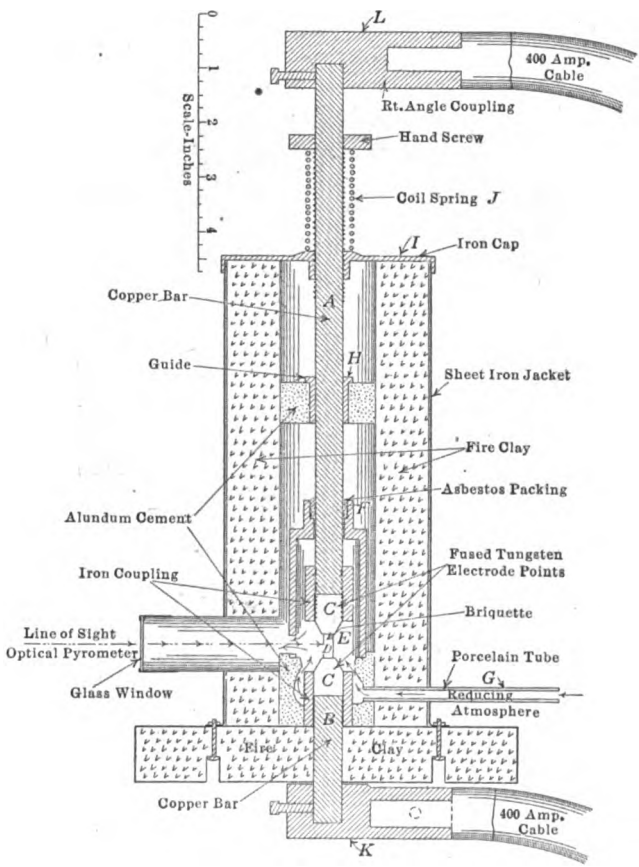

FIG. 20.—ELECTRICAL RESISTANCE FURNACE.

then slowly, as indicated by a bubbler in the purification train. To avoid the fracturing of very fragile briquets, and to prevent a resulting short circuit, the coil spring, *J*, with adjustable collar, is placed so as to support the excess weight of bar and cable.

Fig. 21 is a photograph of this furnace with its door removed so as to show the different parts in place for heating. Fig. 22 shows the complete setup: *A*, being the furnace; *B*, a hydrogen generator; and *C*, an

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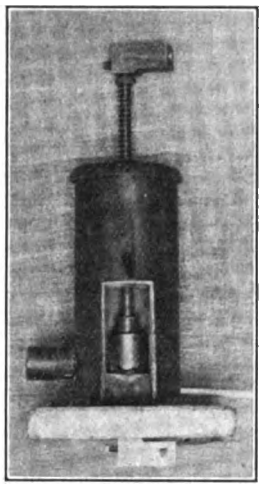

FIG. 21.—ELECTRICAL RESISTANCE FURNACE.

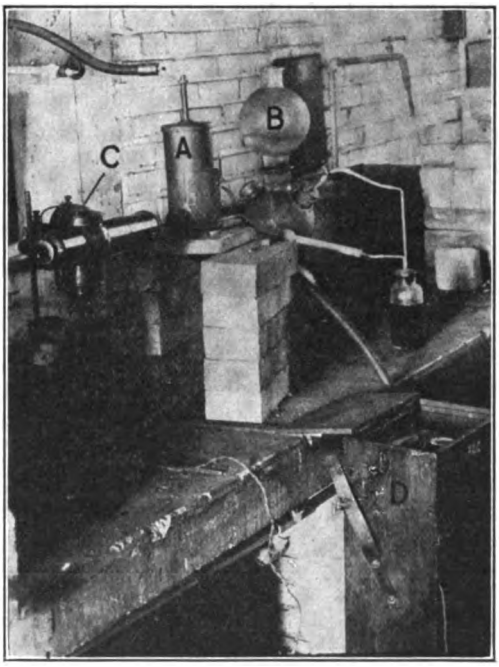

FIG. 22.—ELECTRICAL RESISTANCE FURNACE, WITH CONNECTING APPARATUS.

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optical pyrometer in position for reading the briquet temperature, which is indicated on the scale contained in the case *D*.

Forging

This furnace was designed to be used as a forge, and at the same time to maintain the temperature at which the material of the briquets could be shaped. During this operation the lower bar was firmly supported, while blows were struck with a hammer upon the upper electrode. The increasing of the cross-section and the welding of the material would reduce the resistance considerably, resulting in a lowering of the temperature, unless an increased current were used; but, by pyrometric observation and current regulation during the forging operation, the temperature could be kept fairly constant. The peep hole permitted direct observation of the briquet at all times.

Temperature Measurement.—The conditions under which high temperatures were to be maintained did not permit the consideration of any thermometer or thermocouple method of pyrometry. The practical methods available were those based on the various radiation formulæ, or upon optical observation. A review of this branch of pyrometry revealed an abundance of literature,[20] upon the governing principles of these methods, and their practical application. Of the various types of instrument of this class available, that one was chosen as most practical under the prescribed conditions which involves a direct telescopic comparison of the luminous radiation of the hot body with that of a standard source. There are two makes of this type; one, the Holborn-Kurlbaum,[21] and another, of almost identical construction, the newer type of Morse Thermogage.[22] The working principle is the same in both. A current is passed through the filament of a lamp, which first becomes red and then, with increasing current, successively orange, yellow and white. By interposing this filament between the eye and the hot body to be measured, the current through the lamp may be adjusted until the middle of the filament is of the same color and brightness as the object. At this point the part of the filament used for comparison will disappear against the bright background. The current through the lamp, interpreted in terms of temperature, gives the temperature of the hot body. An absolute match of both color and brightness cannot be made, unless monochromatic light is used, because of the fact that the lamp filament and the viewed body do not radiate similarly. This comparison is facilitated by a

[20] G. K. Burgess: *Measurement of High Temperatures* (A complete treatise and bibliography). See also C. E. Mendenhall and Forsythe, *Physical Review*, vol. iv, p. 163 (1896–7).

[21] Made by Siemens & Halske Aktiengeschellschaft, 94 Markgrafenstrasse, Berlin.

[22] Made by Morse Thermogage Co., Trumansburg, N. Y.

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suitable arrangement of lenses and lamp, as in the instrument of Morse, which was the one used in this work. This is shown diagrammatically in Fig. 23. A low-voltage incandescent lamp, *L*, with M-shaped filament is mounted in the focal plane of the objective, *O*, and the eyepiece, *E*, of a telescope is provided with means of focusing. The lamp circuit is shown connected through a battery, a rheostat, and a milliammeter. Fig. 24 shows this instrument with positions of the rheostat, *A*, and lamp, *B*. Fig. 25 shows a portable case containing battery, milliammeter, and pyrometer.

Temperatures were determined by focusing the instrument upon the briquet so as to bring its image into the plane of the lamp filament, then adjusting the current, by means of the rheostat, until the tip of the filament disappeared against the image of the briquet, when the temperature was read directly from the milliammeter scale (previously calibrated by checking current readings against known temperatures).

Below 1,200°C. readings were made directly, while above this tem-

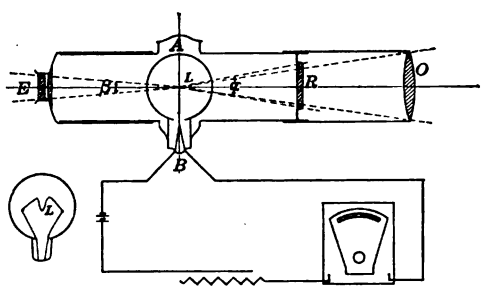

FIG. 23.—DIAGRAM OF MORSE THERMOGAGE.

perature a neutral-colored absorbing "Rauch" glass (*R*, Fig. 23) was interposed between the lamp and the objective, in order to prevent the blinding effect of too great brightness, and also, to avoid overheating the standard filament. A new calibration was necessary when this absorbing screen was used. For temperatures above 1,900°C. it was found necessary to cut down the amount of light entering the pyrometer by means of a diaphragm, with the use of which temperatures up to that of the arc could be read.

The instrument was carefully standardized in all three ranges. The first, in which no absorbing glass was used, was carefully checked against an accurate thermocouple. The second range, using an absorbing light screen, was checked up to 1,375°C. against a thermocouple, and at higher temperatures by the freezing points of palladium and platinum. The Gran-annular electric furnace was used in this standardizing work up to, and including, the melting temperature of platinum, and for this purpose approximated very closely the desired black-body conditions. These standardizations gave current-temperature curves up to nearly 1,800°C.,

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which could be continued by interpolation, with a fair degree of accuracy, up to 2,000°C.

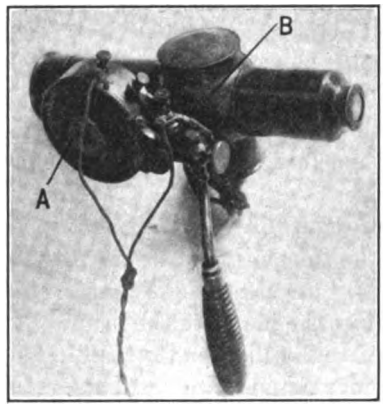

FIG. 24.—MORSE THERMOGAGE, WITH RHEOSTAT AND LAMP.

A third curve was formed by the melting points of platinum, iridium (2,200°C.) molybdenum (2,500°C.) and tungsten (3,000°C.). These latter points were obtained by sighting on electrically heated strips of the

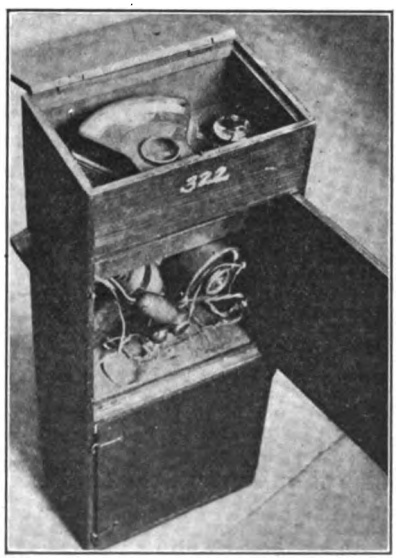

FIG. 25.—PORTABLE CASE, CONTAINING BATTERY, PYROMETER, ETC.

metals formed into narrow wedges, as recommended by Mendenhall,[23] which produces a deep cavity giving very near black-body conditions.

[23] Burgess and Le Chatelier: *Measurement of High Temperatures*, 3d edition, p. 334 (1912).

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The tests on tungsten and molybdenum were made by grinding very thin a portion of a piece of wire, which, on account of the resulting small cross-section, heated very readily. These tungsten and molybdenum strips were arranged as lamp filaments in a glass tube, *in vacuo*. Readings were also made on fusing briquets of these metals, when heated, under operating conditions in the above-described resistance furnace.

A final reading was made on the positive crater of the arc, accepting for this a temperature of 3,600°C. It has been shown that an absolute match of both color and brightness cannot be made, unless monochromatic light is used, or unless the lamp filament and object radiate similarly, which is seldom the case. All observations were, therefore, taken through red screens, which, with a radiation characteristic of about 2,000°C., gave a very narrow band with maximum of transmission at $\lambda = 0.659\mu$.

In all readings care was taken to maintain axial symmetry and constant distance from the hot body to the objective, thus insuring a sufficiently large background, and also constant angles at ∞ and ζ.

The accuracy of this method of pyrometry has been fully proved and is such that between 1,000°C. and 2,000°C. the error of reading should not exceed 10°C. Above this range the error was found to be somewhat greater, but not so great as to interfere in any way with a duplication of desired conditions.

Metallography

Grinding and Polishing.—After heat treatment the ingots were made ready for microscopical examination. The preparation of specimens of this small size (about 0.5 by 0.5 by 1.0) cm. was rather a delicate operation, but the final surfaces secured were all that could be desired. Excellent apparatus was available for this purpose, while the three grades of alundum, F, XF, and 65F, were found to be sufficient for polishing all samples. Care was taken during these operations not to introduce surface conditions which would mask important internal phenomena.

Etching.—For pure tungsten and molybdenum or their high-percentage alloys, the most satisfactory etching reagent was found to be concentrated hydrogen peroxide. It is necessary, however, to have this boiling while in contact with the specimen, otherwise it is quite inert. For the alloys containing a precious metal also, either hydrogen peroxide or aqua regia was used, depending upon whether the tungsten and molybdenum, or the precious metal was to be attacked.

Microscopy.—For microscopical examination, there was available, from both Carl Zeiss and E. Leitz, a very complete equipment, the detailed description of which is not deemed necessary.

Results

In order to test the soundness of the premises upon which the experimental work was based, it was necessary that experiments should first

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be directed toward the production of workable masses of pure tungsten and molybdenum themselves, for, should this not be possible, it would be useless to extend the work to their alloys; and, conversely, should these pure metals yield to such systematic treatment, as based upon conclusions drawn from earlier study of the probable theoretical conditions involved, then the application of similar methods should meet with success in the treatment of their alloys.

It must be pointed out here that no treatment would overcome defects in these alloys due to chemical interactions, but that if, for instance, brittleness, as in the case of tungsten and molybdenum, is due to a purely physical condition, then a method based upon the above considerations should meet with success.

There are two fundamental critical points to be determined: First, the proper ingot-forming temperature; and second, the lowest possible

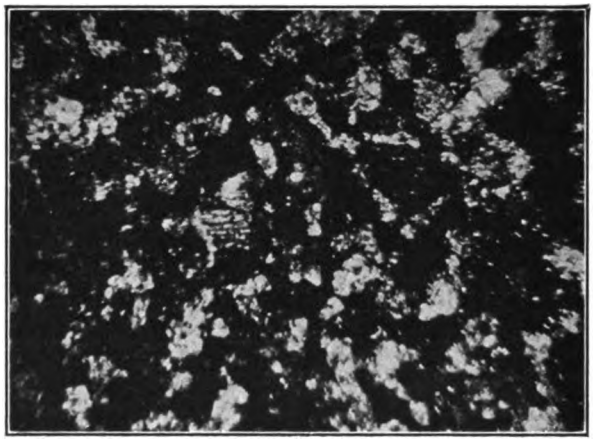

FIG. 26.—RESULT OF LONG, RELATIVELY LOW HEATING.

temperature at which this ingot will yield to mechanical working. This latter point must obviously be below the hot-working range.

With the above considerations in mind, and in accordance with the principles laid down in the preceding pages, the experimental work has been carried out, with results as shown briefly under the following separate heads: (1) Pure tungsten; (2) pure tungsten plus 0.75 per cent. ThO_2, (3) pure molybdenum; (4) tungsten-gold series; (5) tungsten-palladium series; (6) tungsten-molybdenum series.

Pure Tungsten.—The pure tungsten briquets of this series were made under a pressure of 14,500 kg. per square centimeter (about 200,000 lb. per square inch). The specific gravity of the resulting briquet was 13, taken as an average of nine specimens. This was determined from micrometer measurements of volume, and weight of the specimen, and represents about 30 per cent. of void in the briquet.

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From numerous results, the following are taken as typical of the changes which take place upon treating the amorphous briquets at various temperatures:

TABLE III.—*Results of Heat Treatment of Briquets*

Number of Specimen	Treating Temperatures	Time of Heating		Remarks
	Deg. C.	Hr.	Min.	
5	500	1	0	Fairly coherent. Partly sintered. Could not be polished.
6	600	1	0	Harder and stronger than No. 5. Could be poorly polished.
8	800	1	0	Hard and metallic. Filed with difficulty. Polished readily.
10	1000	1	0	Very hard and metallic. Polished to beautiful surface.
12	1200	1	0	Appeared microscopically as solid metal. Very hard and brittle.
14	1400	1	0	As No. 12. No crystal formation.
16	1600	1	0	An apparent internal change, which could not, however, be called crystalline.
18	1800	1	0	Showed changes indicating crystal formation, but with no definite outlines. (Gran-Annular furnace.)
22	2200	0	10	Appeared about as No. 18. (Briquet resistance furnace.)
26	2600	0	10	Very finely crystalline with many intercrystalline boundaries not clearly defined—indicating presence of amorphous material.
27	2700	0	5	More coarsely crystalline than No. 26 with crystals in center larger than at sides.
28	2800	0	1	About as coarsely crystalline at surface as No. 28, but area in center indicated a complete liquid fusion.

The extent and type of crystallization were found to depend upon the time-temperature factor. Prolonged heating at relatively low temperatures gave fewer and larger crystals, and crystals of different sizes (Fig. 26), while a flashing at higher temperatures (specimen No. 26) gave a very fine and uniform crystalline structure (Fig. 27), which was much to be preferred. In specimen No. 28 (Figs. 28 and 29), the heat of radiation from the sides of the briquet was so great that while the surface-temperature reading was only 2,800°C., the molten interior indicated a temperature above the melting point of tungsten (3,000°C.) at the center. Photomicrographs, Figs. 30 to 32, show the transition from the compressed powder to a solid, metallic, apparently non-crystalline mass, and finally to a solid, crystalline ingot.

The next step was to determine which of these stages of crystallization gave the best forging conditions. This was done by treating a series of each of these numbers at varying temperatures. It was found that,

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within certain limits, the lower the sintering temperature, the higher was the initial temperature necessary for forging.

No. 5 could be forged at about 2,400°C.; No. 10 at about 2,200°C.; No.

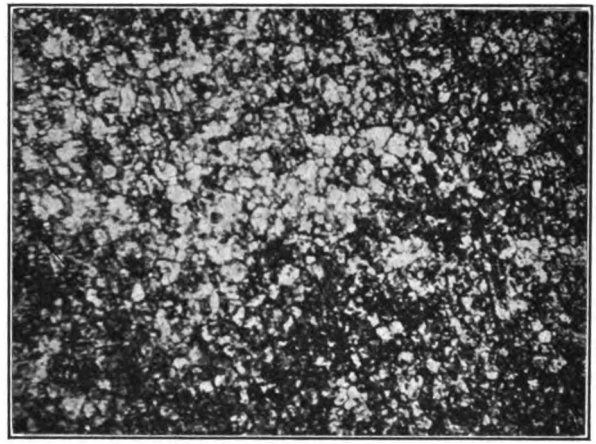

FIG. 27.—RESULT OF QUICK HIGH HEATING.

18 at about 2,000°C.; No. 22, at about 1,700°C.; No. 26, at 1,275°C.; No. 27, at 1,350°C.; No. 28, at 1,375°C. These figures, of course, represent the initial welding temperature of the more or less powdery mass, the

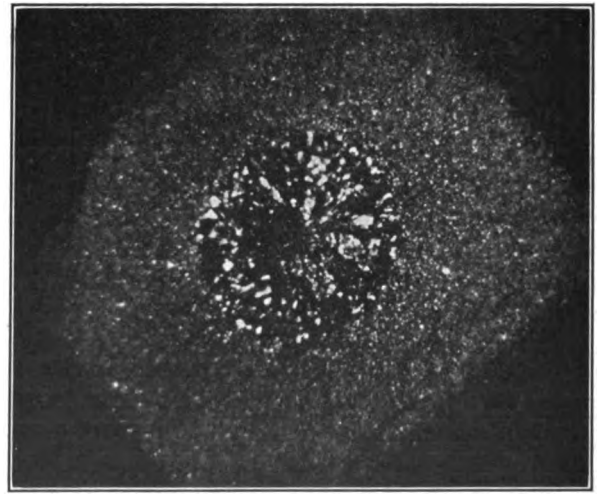

FIG. 28.—MOLTEN INTERIOR, SPECIMEN 28.

necessary temperature falling at once when this welding had begun; they give no definite information other than indicating the necessity for at least partial crystallization in the treated ingot. Below these high

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temperatures the specimens were brittle in the first stages of forging, and if these temperatures were maintained after forging began, there resulted a marked crystal growth, instead of the desired reduction.

These series were represented by 92 forgings, or attempted forgings,

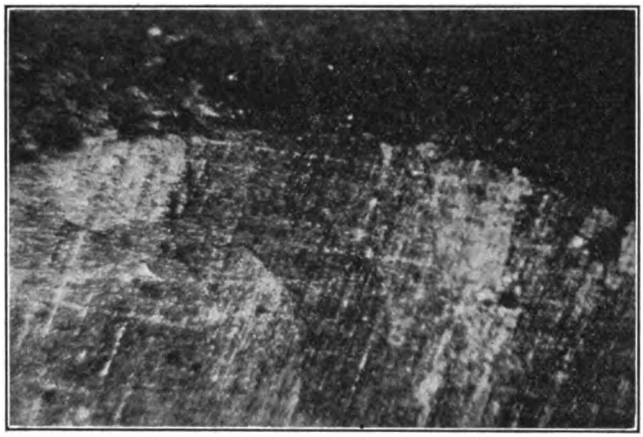

FIG. 29.—SURFACE, SPECIMEN 28.

and the results gave a curve with a minimum forging temperature corresponding to a certain crystal size. This is represented approximately by No. 26 (micrograph Fig. 27). A higher temperature for a shorter time would give nearly the same conditions, or a lower temperature for a longer

FIG. 30.—FIRST STAGE, COMPRESSED POWDER.

period. It is thus seen that it would be useless to prescribe a definite treating temperature, or even one for forging, because the temperatures recorded were observed under rather restricted conditions, and in apparatus giving the worst possible conditions as to radiation, uniform heating, etc., due to the small scale on which the operations were carried out.

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The results, however, showed how dependent are the physical properties of wrought tungsten upon the heat treatment and the crystallographic control exercised during the various stages of its manufacture.

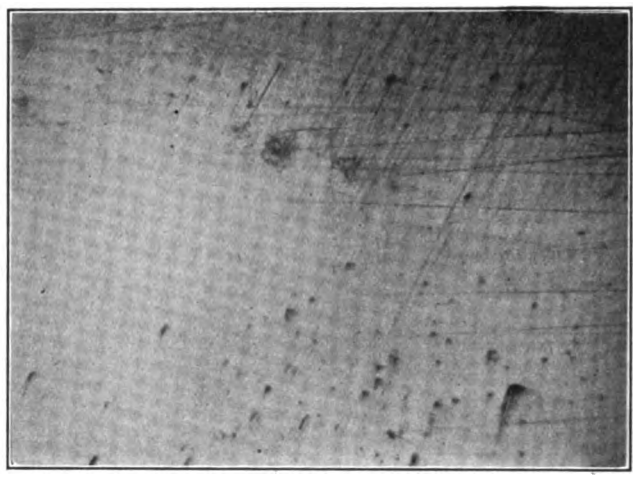

FIG. 31.—SECOND STAGE, NONCRYSTALLINE MASS.

The best specimens of this series were hammered out to a thickness of less than 0.5 mm. and, although it was necessary to increase greatly the heating current with increased cross-section of the ingot, the contact

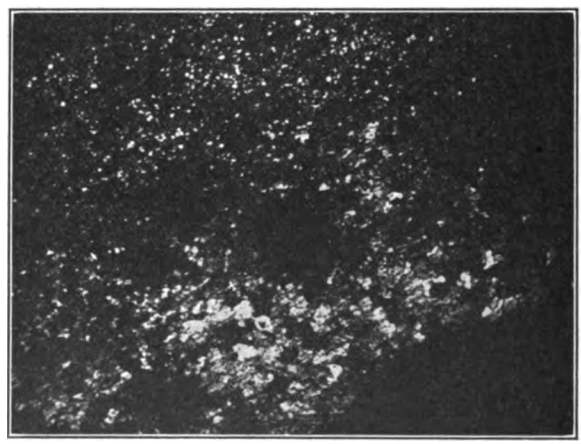

FIG. 32.—THIRD STAGE, SOLID CRYSTALLINE INGOT.

resistance was enough to maintain a sufficiently high temperature. As the electrodes approached each other it became increasingly difficult to make accurate temperature measurements, since only the irregular outside edge of the flattened ingot could be sighted upon with the pyrometer

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No means were available with which to draw this material, so that its behavior during that operation, and the influence of the above-described control can only be assumed.

Tungsten with 0.75 *per cent. Thorium Oxide.*—An exactly similar series of experiments revealed only one point in which tungsten and thorium oxide reacted differently from the pure tungsten; but that point is of the greatest importance.

Sintering progressed with the gradual appearance of crystallites in a manner so closely paralleling the preceding case that the presentation of another almost identical set of figures and micrographs is thought unnecessary. But with respect to the crystal growth at higher temperatures, there was a marked difference. The presence of this small amount of thorium oxide was found largely to prevent a coarsely crystalline formation in highly overheated specimens; and even at stages just below the actual melting point the structure was very uniform. The explanation for this marked influence of thorium oxide is probably as follows:

Crystallization of this powdery, amorphous mass no doubt begins, as in the case of crystallization from the liquid state, at numerous centers, and proceeds, as in that state also, by the impressed orientation and resulting "absorption" of adjacent molecules of amorphous material, accompanied by an expulsion of foreign matter outward until, when neighboring crystals meet, the intercrystalline boundaries consist of thin sheets of this foreign matter. In most known cases the presence of this foreign material produces a weakness in the entire mass, but in the case of tungsten, the thorium oxide seems to be in such form as not only to prevent crystal growth, by its presence among the molecules of tungsten and later, between adjacent crystallites, but to maintain the strength of the entire mass. This may be a phenomenon similar to that which may be caused by iron oxide when two pieces of iron are welded together.[24] It has been noticed, in this case, that no matter how great the crystal growth occurring within either of the welded parts, no growth will extend across the weld; *and this weld is usually the strongest section of the specimen.*

Several experiments were performed, in which the thorium oxide was replaced by other refractory oxides. Magnesium oxide, cerium oxide, and other refractory earths, produced a similar retarding action upon crystallization, but the effect was not so marked. Although the experiments on other oxides than thoria were not sufficiently extended to locate these materials on a scale of their comparative effects in retarding the crystallization of tungsten and molybdenum, their value, for this

[24] It might be of interest to determine the effect of iron oxide upon the crystallization of iron, and upon its physical properties, by adopting a method similar to that here used for tungsten.

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purpose, seems to be a function of the temperatures at which they begin to soften.

The explanation of the accumulation of thorium oxide at the intercrystalline boundaries was first given by Zay Jeffries;[25] and this phenomenon has been observed so often, during studies in connection with these experiments, that the logic of the explanation seems proved. Nothing is known regarding the physical characteristics of this rare earth when segregated under these conditions, but there is no reason for believing that it may not be in a vitreous, amorphous form, and hence, of great strength.

The influence of thorium oxide during forging was not so marked, but this is due to the fact that these experiments were carried out only to such extent as to prove that this material could be forged under the above condition. Indirectly, however, the effect of thorium oxide was present, in

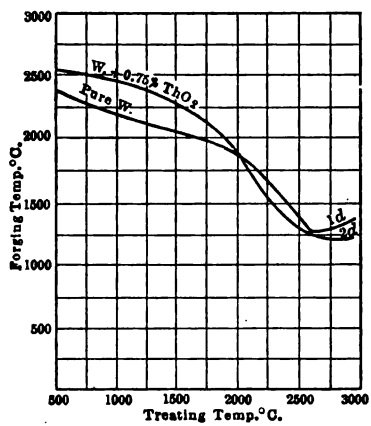

FIG. 33.—TREATING AND FORGING TEMPERATURES FOR PURE TUNGSTEN AND TUNGSTEN CONTAINING 0.75 PER CENT. OF THORIUM OXIDE.

that those ingots formed at the higher temperatures, and in which excessive crystallization had been prevented by its presence, could be forged at the lower temperature, the curve not showing an upturned branch at the high-treating temperature end, as had that for pure tungsten (see Fig. 33).

Pure Molybdenum.—The material used in this series, obtained from the General Electric Co., was the chemically pure powdered metal used in the manufacture of drawn wire.

The raw briquets of molybdenum were made under a pressure of 14,500 kg. per square centimeter and gave an average specific gravity of 7.80. Table IV shows the change in internal structure of briquet with increasing temperatures.

[25] Private communication, 1914.

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TABLE IV.—*Structural Changes in Briquets of Molybdenum*

Specimen Number	Temperature, Deg. C.	Time, Hr.	Time, Min.	Remarks
4	400	1	0	Slightly harder and more coherent than raw briquet.
8	800	1	0	Hard enough to be poorly polished.
12	1,200	1	0	Very hard and brittle, but of considerable compressive strength. Polished as solid metal.
16	1,600	1	0	Stronger and more metallic than No. 12.
20	2,000	1	0	Structure changed. Showed patches which microscopically behaved as solid crystalline material, but with no definite boundaries.
22	2,200	1	0	Coarsely crystalline, but with grains of very irregular size.
24c	2,400	0	10	Coarsely crystalline, grains of more regular size than 22.
23g	2,300	0	1	Very finely crystalline. Ingot quite homogeneous throughout, best structure of series.

The photomicrographs of this series could not be distinguished, in representing the transition from compressed amorphous powder to crystalline ingot, from those for pure tungsten, and hence are not given.

Specimen No. 23g, was of minimum grain size consistent with maximum crystalline formation. In this, as in the preceding series, it seemed necessary to reach a certain size of grain in order to produce an ingot free from patches of incompact amorphous material, the presence of which made it necessary to use a very high temperature at the beginning of the forging operation. On the other hand, too high a treating temperature produced a coarsely crystalline structure, with its inherent brittleness, which also could not be overcome, except at high temperatures.

The forging curve for pure molybdenum showed a minimum corresponding to No. 23g, with higher temperatures on either side, as in the case of pure tungsten.

The results of experiments on tungsten and molybdenum have given excellent support to the theory upon which the work was based, and furnished details of critical points which promise to be paralleled, and should be searched for, in any series of their alloys.

Having established the soundness of the principles involved when applied to these pure metals, the next step was to apply them in connection with their alloys.

It must be again pointed out that this method did not have as its object the production of true alloys, in the sense of those which solidify from the liquid state, but to avoid such consequences as might result from

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that condition, by welding together, at a temperature below the melting point, particles of molecular proportions, which should be as evenly distributed and uniformly mixed as are the molecules in a true solid solution.

Tungsten-Gold Series.—To insure an intimate mixture of gold and tungsten the chemically pure tungsten metal was first oxidized and then saturated in an evaporating dish with an acid gold chloride solution containing the desired percentage of gold. This was carefully evaporated and dried, with constant stirring to prevent segregation; then transferred to an alundum boat and placed in the quartz-tube furnace for reduction under pure hydrogen. This material, when compressed into briquets, under 19,300 kg. per square centimeter (about 265,000 lb. per

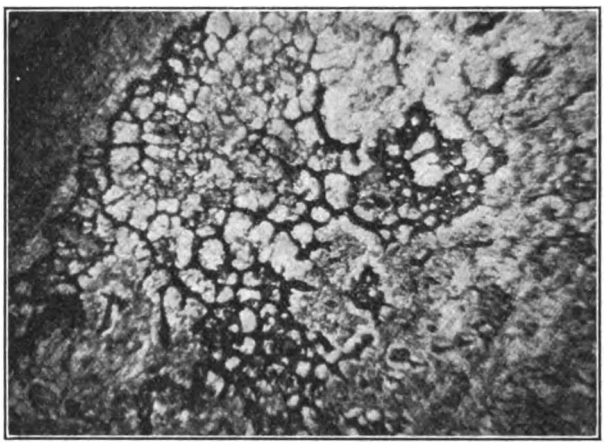

FIG. 34.—TUNGSTEN-GOLD MIXTURE, CONTAINING MORE THAN 1.4 PER CENT. OF GOLD. DARK AREAS, GOLD. ETCHED WITH AQUA REGIA.

square inch) pressure and examined under 2,500 diameters, did not show the slightest non-uniformity.

This series was investigated through a range from 0.1 to 20 per cent. of gold; but to give in detail the numerous negative results obtained would make this paper too cumbersome. A brief description of the behavior of these combinations in several proportions will suffice for the series.

No combination in this series could be forged at any temperature; even 0.1 (which was the smallest percentage of gold used) seemed to be sufficient to prevent working of the mass.

Below 1.4 per cent. the gold could not be detected as segregated at any stage of the treatment. This amount, however, was sufficient to cause coarse crystallization at about 2,500°C. Analysis showed the rather remarkable fact that very little of this gold was lost, even in those specimens which were treated to a temperature of 2,600°C., which is far above the accepted boiling point of gold.

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The extreme fragility of these specimens at very high temperature prevented even an approximate determination of the melting point.

Above 1.4 per cent. of gold, a segregation resulted in those specimens treated between about 1,200°C. and 2,200°C. (see Fig. 34). Below about 1,200°C. no change could be detected microscopically, while above 2,200°C. a homogeneous crystalline mass resulted. An interesting phenomenon is noted in connection with temperatures above this latter point. In the case of specimens containing 5, 7, and 10 per cent. respectively of gold, after being heated to a temperature near 2,600°C. the gold content was found to be partly volatilized, and condensed in visible particles on the chamber walls and on the electrodes. In each case there remained a coarsely crystalline mass containing from 4 to 5 per cent. of gold.

No attempt was made to learn the nature of this material or to determine its properties, other than to prove its worthlessness from a mechanical point of view.[26]

Tungsten-Palladium.—A similar investigation of the tungsten-palladium series, but involving only 0.0 to 4.5 per cent. of palladium, gave results very similar to those of the preceding series. Below the melting point of palladium, no change could be detected. Above this point the mass became crystalline and possessed an inherent brittleness which could not be overcome. This was no doubt due to the fact that some reaction had taken place. Whether a compound or solution was formed is not known, but that combination of some kind had occurred, was attested by the fact that the mass appeared homogeneous at all stages of the treatment under the highest magnification.

These ingots of tungsten containing palladium were not found to be malleable, when worked under the above-described conditions.

Tungsten-Molybdenum Series.—The tungsten-molybdenum series was investigated in various proportions of the components, taken at 10

[26] It would be interesting to try this method upon combinations of metals such as nickel and palladium, copper and gold, or a similar series, whose components were of nearly equal melting point; for the failure to produce results in the case of gold and palladium with tungsten may be ascribed to a too great difference between melting points of the components. Near the lower range, the tungsten is not sufficiently plastic to permit a welding between the particles; while at higher temperatures, fusion conditions are present.

Even the pure metals, as gold, copper, iron, etc., might, if prepared in this way, possess properties far superior to those of the cast and worked metals, for in this manner the mass may be made to consist more completely of amorphous material, with its attendant characteristics, as shown in the results of experiments on pure gold, as described above. Even though an entirely amorphous condition could not be maintained throughout the operation, at least a much more finely crystalline ingot might result than from any available fusion method.

Experiments have been started with the view of testing this out, but have not progressed to that stage which would permit a definite statement regarding this possibility.

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per cent. intervals through the range from pure tungsten to pure molybdenum. The alloys represented by each of these intervals were found to be susceptible to certain conditions under which they could be wrought. The briquets were made under a pressure of 18,000 kg. per square centimeter and were then treated through a range of temperature in a manner similar to that employed in the previous series. Figures for the treating and forging temperatures of pure tungsten and molybdenum have already been given, while those for intermediate points are shown on the curves, Fig. 35.

The investigation of this series involved the treatment of more than 200 small ingots, and to give the details regarding each would be merely to repeat those already given for pure tungsten, which may be taken as

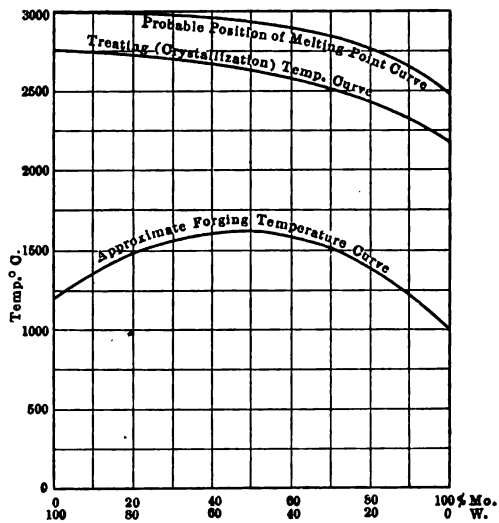

FIG. 35.—TREATING AND FORGING TEMPERATURES OF TUNGSTEN-MOLYBDENUM SERIES.

typical for any of these series. The formation of crystallites and crystal growth proceeds in a manner shown in the photomicrographs of the pure tungsten briquets, although—and here is an important point—the temperature representing a certain grain size changes with the relative amounts of the components, as the curve clearly shows. This is to be expected, and must be observed, in the treatment of the ingots; but it is not known whether this factor has received consideration in any attempted commercial production of such alloys.

A factor of equal importance is that of a corresponding change in the temperature of forging. If, for instance, only 1 per cent. of molybdenum were added to the tungsten, a treatment identical with that for pure tungsten would produce grains of such large size that even after repeated

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forging (and this requiring a higher relative temperature) the material would still be unreliable. On the other hand, the addition of small amounts of tungsten to molybdenum would necessitate higher treating and forging temperatures, as shown in Fig. 35.

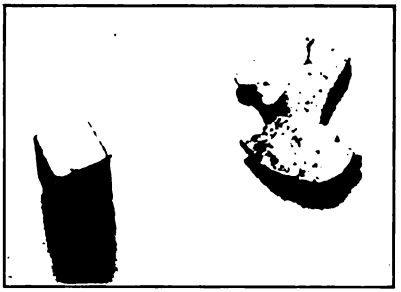

FIG. 36.—RAW AND FORGED BRIQUET OF 50 PER CENT. TUNGSTEN-MOLYBDENUM ALLOY.

Microscopical examination of these treated specimens revealed nothing that would indicate that these two metals formed other than a complete series of solid solutions; and if the crystallization curve may be taken as paralleling the melting-point curve (which assumption seems

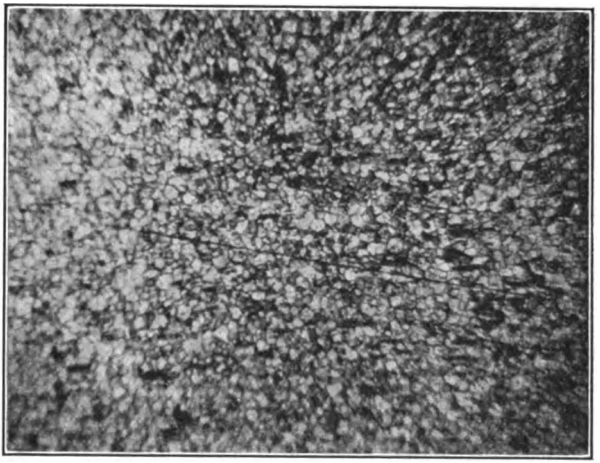

FIG. 37.—MICROSECTION OF BRIQUET, FIG. 36, AFTER TREATMENT AND BEFORE FORGING.

not illogical), then this latter curve will be found, when determined, to occupy a position approximately as indicated in Fig. 35.

Fig. 36 shows a specimen of the 50 per cent. alloy as the raw briquet and after being forged; while Fig. 37 shows a microsection of this same specimen, after treating and before forging.

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VI. Summary and Conclusions

With regard to the degree of accuracy with which temperatures could be measured in these experiments; it must be pointed out that the object was not to establish these critical points for direct transference to any commercial plant (for different types of apparatus would necessitate a determination of these conditions to suit each individual case), but to determine their existence and influence. It would also be of no avail to locate these critical ranges because every different set of apparatus and conditions would require a new standardization.

In these experiments, however, the temperatures necessary to produce a certain degree of crystallization were considered as being located with a fair degree of accuracy, insofar as this may not be qualified by the existence of working conditions which were far from ideal.

As to the measurement of forging temperatures, no claim is made for more than close approximations, for this was properly not a one-man operation, and was performed by the writer with one eye to the optical pyrometer, the other on the milliammeter scale; one hand on the pyrometer rheostats, the other using a hammer on the upper electrode; while the heating current was controlled by one foot on a lever regulating the transformer and rheostat.

The experimental work resolved itself into three parts, each being marked by a different method of attack, necessitated by limitations encountered as the work progressed under previously adopted methods.

The first part consisted of experiments on binary combinations of those of the metals which it was feasible to consider, and the melting points of which lay within the limits of ordinary fusion methods. The results of these experiments, performed as indicated therein, lead to the conclusion that metals or alloys of metals outside of the precious-metal groups are unsuitable as substitutes for platinum.

The gold and silver alloys of palladium have been found to be excellent substitutes for platinum in its softer forms, and while not so chemically resistant, fill all requirements where conditions are not too rigid.

The second part develops the fact that except in two respects, pure ductile tungsten, and, to a lesser degree, molybdenum, meet all of the specifications of a practical substitute for platinum and its alloys. These two defects are its ease of oxidation, and the difficulty with which it can be soldered; and they have been overcome by coating with a precious metal or alloy, the resulting material being in many ways far superior to platinum or its alloys.

This material has met with instant demand, is in many cases replacing the best platinum-iridium alloys, and permits the performance of work which has been impossible with the materials hitherto available.

The third part describes the theoretical and practical considerations

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involved in the manufacture of wrought tungsten and molybdenum, and gives results of the proper application of a similar method in the laboratory production of their alloys.

Wrought tungsten and molybdenum were produced on a laboratory scale, but no success attended the attempted production of alloys of tungsten with gold and palladium; while on the other hand, the alloys of the tungsten-molybdenum series were produced in wrought form. These operations were governed entirely by metallographic control, and their success suggests the possible application of a similar method in a treatment of such metals as iridium, tantalum, rhodium, osmium, etc., in combination with each other, or with tungsten or molybdenum, which may result in the production of alloys possessing properties far superior to those of any material now available.

Discussion

F. A. Fahrenwald, Cleveland, Ohio (communication to the Secretary*).—Since the publication of my paper in the January *Bulletin*, (which was largely a metallographic treatise), I have been questioned regarding two points referred to therein.

One of these is that which deals with the application I have made, (p. 557) of the equation for melting pressures which correspond with different working temperatures. The other refers to the statement, (p. 575) that "Prolonged heating at relatively low temperatures gave fewer and larger crystals and crystals of different sizes, while a flashing at higher temperatures gave a very fine and uniform crystalline structure, which was much to be preferred."

Especial emphasis was not placed upon either of these two factors, for each seemed perfectly logical and obvious. As they do not seem to be self-explanatory, I will supplement with those portions of my notes which were purposely omitted from the manuscript submitted for publication.

A few practical illustrations may serve as evidence in support of the first of these, but it must be understood that in this work the "amorphous theory" is considered as being sound, and that it is assumed that metals allow of distortion, without fracture, by virtue of the formation of slip surfaces, accompanied by the transformation of crystalline material to the amorphous phase.

Effect of Pressure.—In all forging, rolling, and drawing operations, it is known that specimens of small cross-section may be worked at a lower temperature than can be used for the same material in much greater mass and sectional area. In the larger pieces the maximum pressure is brought to bear upon the material near the surface, so that the interior is not subjected to great pressure, but to a stress more

* Received Jan. 15, 1916.

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nearly tensile, or shearing, so only the surface is subjected to the critical melting pressure. In the case of wires which draw hollow, the center is subjected to practically a tensile stress which breaks it, while the surface skin draws perfectly, due to the enormous "flowing" pressure exerted upon it by the surface of the draw-plate.

Also, certain metals and alloys in wire form may be drawn at ordinary temperatures, while these same substances in large ingot form will permit of a proportional reduction of area only at a relatively high temperature.

In the case of tungsten or molybdenum, the decrease of forging and drawing temperature with decrease in cross-section may be explained in the same manner. In the smaller sizes each unit throughout the mass receives a nearly equal pressure and, what is of greater importance, the *unit pressure is greater*, so that a lower temperature value will satisfy the equation in which both pressure and temperature are factors.

Effect of Ingot-Forming Temperature.—Relative to the seemingly anomalous formation, during the ingot-forming heat-treatment of compressed molybdenum powder, of small crystals at high temperatures (Fig. 27) and large crystals at lower ranges (Fig. 26), it would seem that the different time of heating would partly account for this. It is evident, too, that the more intense heat for the shorter time results in a condition similar to that produced by sudden cooling from the molten condition, in which case a large number of uniform grains is formed, and these are in a state of attractional, or orientational, equilibrium.

If the original powder contained the nuclei for crystallization, there was an equal number in each briquet, and they were similarly distributed. They may not, however, have possessed the same orientating force upon the surrounding molecules. For instance, some nuclei may have been made up of a single unit of force (or molecules, etc., as the case may be); others of two or more, and others of several hundred. In the case of the quick high heat, crystallization proceeded so rapidly and with such great ease that a slight handicap possessed by the smaller crystallites was not noticeable in their race outward, and so resulted in a structure in which each nucleus was approximately at the same distance from the surrounding cell wall or intercrystalline boundary.

On the other hand, when the treating temperature was lower and more prolonged, or increased more gradually, the most powerful nuclei exerted their force first and in an exaggerated manner, while the more minute forces could not influence a single molecule.

It is no doubt true that a grain or crystal will have an orientating influence in direct proportion to the number of units which it already possesses, so that if a molecule or crystallite lies equidistant between a grain possessing 100 units and one possessing 101 units, it will, other

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conditions being equal, be oriented parallel to the larger mass. In the case of the coarse-grained, low-temperature, specimen (Fig. 26) it may be that the molecular freedom of the mass was such that a nucleus would have to possess 10 or more (or any definite number *distinct for that particular temperature*) of units in order to impress its orientation upon a single molecule. At a slightly higher temperature a nucleus of nine or $x - 1$ units could grow, and still higher, one of seven, or less or $x - n$, or whatever force was necessary to overcome the viscosity of the mass, or the opposing attractions, at that particular temperature.

In this argument it is assumed that the reduced molybdenum powder was in practically a molecular condition, or at least one of very small aggregates, in which case it seems that those below a certain force or number of units, depending on the temperature, are swallowed by those above this line.

After the mass is completely crystallized, unless this has been accompanied by a cast-metal type of equilibrium, these contending forces will continue to battle for supremacy, and at any given temperature a distinct excess of force represented by molecular vibration, will be necessary before one grain can impress its orientation upon its smaller neighbor. The higher the temperature, the smaller will that force need to be and *vice versa* due to a greater molecular freedom.

These contending forces may be likened to that existing between two bubbles of different sizes which have their interiors connected through a tube. In this case one will "absorb" the other in conformity with a law of nature. If the bubbles were of exactly equal size there would be equilibrium. So in the case of metallic masses; unless there is perfect equilibrium between contending orientating forces, grain growth proceeds and non-uniformity increases. In the case of cast metals, this equilibrium accompanies crystal formation so that by the time the material is completely solid all opposing forces have been balanced, and as a result reheating below the melting point can cause no further rearrangement in metals which have not been distorted.

The apparent discrepancy referred to is explained then by considering that in the case of small grains at high temperatures (Fig. 27) the growth from every nucleus commenced simultaneously and proceeded at an equal axial rate until the intercrystalline boundaries were reached. If the temperature was high enough this would be accompanied by a cast-metal type of equilibrium, while in the case of the coarse and non-uniform specimen (Fig. 26) the unbalanced forces were given time to act and conditions under which they would become exaggerated. If no nuclei were present, *i.e.*, if the material was entirely amorphous, the quick high temperature would force a great number simultaneously, while at the lower temperature only a few would be generated, and their conditions of growth would proceed as in the above case.

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In case a compressed metal briquet was so heated that a given area was traversed by a temperature gradient, *i.e.*, if it was hotter in the center than at the edge, as would be the case in electrically heated specimens, grain growth would begin at that area which was at a temperature favorable to equal growth; for the hotter part would crystallize almost simultaneously, with an accompanying orientational equilibrium similar to that of a cast metal; while the cooler part would not possess sufficient molecular freedom to increase the size of a single grain. At the junction of these two areas, however, the more powerful nuclei would grow at the expense of the smaller, and with comparatively large grains once formed this area would encroach rapidly upon the hotter area—which of itself was in approximate equilibrium—and to a certain extent, and more slowly, upon the cooler portion.

This would be true, however, only in case the specimen was brought very quickly to the maximum treating temperature, for with a gradual heat the passage of the final hotter portions through this range will have started unequal growth, which only becomes exaggerated upon raising the temperature.

With these facts revealed it became necessary, in order to produce a uniformly fine-grained briquet, only to heat *very rapidly* to a point well above the normal crystallizing range, in fact near the melting point of the metal.

HENRY M. HOWE, Bedford Hills, N. Y. (communication to the Secretary*).—Do I understand aright the author of this very valuable paper as meaning that the support for the "Amorphous Cement" theory referred to by him on p. 558 is the certainly extremely great increase of the cohesion of gold with increasing grain fineness? If so, I would point out that this supposed intergranular amorphous cement is only one of four explanations offered of this increase, and ask him what reason he finds for holding that this increase supports the amorphous cement theory rather than any of the three others? I have elsewhere[27] elaborated my reasons for regarding the boundary strength as due primarily to the progressive accumulation of amorphous metal along the grain boundaries during the deformation itself, because of the disregistry between the slip planes of adjoining grains.

It is not clear to me that the fact that the increase of cohesion with increasing grain fineness in the Author's results is much greater than in the cases which I have met before really bears on the cause of the boundary strength.

The strengthening caused by plastic deformation is reasonably referred, at least in part, to the formation of amorphous metal along the

* Received April, 1916.

[27] *The Metallography of Steel and Cast Iron*, p. 506.

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slip planes, but that in itself does not really imply, if it even suggests, that the boundary strength also is due to amorphous metal.

F. A. Fahrenwald (communication to the Secretary*).—The results of experiments to which Professor Howe refers were offered more in support of the contention that amorphous metal is actually present at intergranular surfaces, than as an explanation of the manner in which this phase was caused to exist.

An analysis of conditions under which these test specimens were prepared, however, indicates that the results shown here are stronger in their support of the presence of amorphous metal which has been formed in accordance with the hypothesis of Prof. Howe, than they are in their support of any theory assuming the pre-existence of amorphous boundary filling material.

In the case of each of the specimens (2), (3), and (4), the individual particles were subjected to a type of deformation which tended to form amorphous metal at the particle boundaries. In No. 2 the gold filings were subjected, during preparation, to considerable surface rubbing in addition to the deformation produced by cutting and shearing. The shape and surface condition of the individual particles making up specimens (3) and (4) are not known.

By tracing this material, now, through the briquetting process, it is observed that an unusual type of plastic deformation plays an important part. When packed loosely in the mold, considerable more than the theoretical amount of void exists, due to the fact that in this porous mass, only points and irregular surfaces are in contact. As pressure is applied a shearing action is set up within each particle and, of greater importance, a rubbing and gouging of one surface upon another is induced by the progressive application of pressure until the actual contact[28] of all intergranular surfaces is accomplished.

The greater the state of subdivision the more amorphous surface material was produced, due to greater surface area. The internal shearing action was also more intense in the case of the finely divided metal, due to the presence of a smaller number of crystalline units within each particle. The large and fine particles underwent the same relative deformation during the elimination of void spaces; the former by a large number of small displacements and the latter by a smaller number of relatively larger slips.

There can be very little doubt of the fact that the interfacial amor-

* Received April 21, 1916.

[28] This condition is not reached, however, unless a vacuum is employed during compression. A later heating of these specimens caused a marked swelling, due, no doubt, to imprisoned air. The nature of its distribution within the compressed briquet is not known. This phenomenon was not noticed in the case of harder and more brittle metals.

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phous material was, in this case to a certain extent at least, produced by the breaking up of crystalline metal. This slipping under pressure of one surface upon another may have produced a condition similar to that found at slip surfaces within the grains, or at the intergranular surfaces as suggested by Prof. Howe (4 below). The micro-examination of surfaces of rupture disclosed nothing which indicated that an actual pressure fusion had taken place. These surfaces did not resemble an ordinary metallic fracture, and showed no lines of demarcation separating the individual particles. The surface bore a rubbed, greasy, appearance quite unlike any that I have elsewhere observed.

I take the liberty to quote the four explanations of the strength of intercrystalline boundaries, as outlined by Prof. Howe.[29] They are as follows:

(A) Three supposed to exist before deformation, viz.:

1. Strong amorphous boundary filling (Osmond, Bengough, Rosenhain, *et al*).
2. Dendritic interlacing of crystals across boundaries (Rosenhain).
3. Progressive confusion of crystalline structure as the boundaries are approached (Humfrey).

(B) One supposed to be developed during deformation, viz.:

4. Progressive accumulation of amorphous metal about the boundaries through the break-up of crystalline metal caused by disregistry of the slip planes in the adjoining grains (Howe).

The manner of preparation of the test specimens under consideration could produce either (3) or (4). Certainly (2) is eliminated, and (1) could exist only within the mass of each particle.

The results of these experiments on briquetted gold particles are evidence only in support of the hypothesis that increased strength is due to the formation of amorphous material, in that they show the converse of this condition to be true, *i.e.*, that increased rigidity accompanies increased amorphous metal content.

It is not clear to me, however, that (4) can be a factor in accounting for the greater *initial* strength of fine-grained cast or equiaxed metals when compared with that of these same metals in a coarse-grained state. Explanation (4) does not enter into consideration until the elastic limit has been exceeded, so that the *initial* strength of intercrystalline boundaries is dependent upon amorphous boundary material already present.

The intergranular fracture (of Rosenhain, Ewen, *et al*) in metals at temperatures just below their melting points is strong evidence in support of the existence of this amorphous cement. I cannot, at this time, conceive of a "thin film of a weak or brittle intergranular alloy, possibly of the metal with some gas undetected by analysis."[30] for upon heat

[29] H. M. Howe: *Metallography of Steel and Cast Iron*, p. 496 *seq.*

[30] H. M. Howe: *Op. cit.*, p. 498.

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treating with a resulting decrease or increase in grain size, it is not apparent to me what becomes of this intergranular alloy. In the case of grain-growth especially, should not the decreasing intergranular surface fail to accommodate the constant amount of intergranular alloy, and so produce a segregation of this material?

Further evidence in support of the "amorphous cement" theory, may be added in an affirmative answer to Prof. Howe's tentative statement[31] that "Moreover, if the crystal units are of appreciable size, then fine-grained metal, because it has more boundaries, should be lighter than coarse-grained."

I have observed the effect of this condition during the investigation of several industrial problems, and in cases where temperatures are encountered which permit of grain-growth, it is of very great practical importance. One example of this was so striking that I regret the impossibility of including detailed observations at this late date. Results of interest in connection with the question of grain boundaries will be given in the future. In brief, the circumstance was as follows: In the commercial application of a certain alloy wire, working temperatures varying from 20°C. to 85°C. were encountered; the latter being well above the equiaxing temperature for this alloy. Under these conditions many failures occurred, some after a period of a few hours, others after a month or more. An investigation of the causes for this failure revealed the following condition: The wire had been cold drawn and annealed at the manufacturers under such conditions that its structure was coarsely-crystalline near the axis and very fine-grained near the surface. This was due to the fact that the surface material during drawing had been subjected to a much greater relative deformation than had that near the axis, while the subsequent annealing temperature was such that the above condition resulted.

When employed at high temperatures, the large grains grew from the axis outward by absorbing the smaller grains until, when the material became completely coarse-grained, the decrease in intergranular surface, and hence in the less dense amorphous metal, became so great that the grains near the surface actually pulled apart. This was due to the fact that the material near the surface of the specimen had decreased in total volume, due to a transformation of pre-existing, intergranular, amorphous material to the crystalline state; while the volune, or length, of the axis remained constant. These cracks, with fluctuation of temperature, soon worked their way entirely across the specimen, with resulting failure.

It is entirely possible that some other explanation may be given of this condition, but it is only one of many examples that I have encountered which causes me to give credence to the intergranular "amorphous cement" theory.

[31] Howe: *Metallography of Steel and Cast Iron*, p. 496 (No. 696).

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The Determination of Grain Size in Metals*

BY ZAY JEFFRIES,† B. S., MET. E., A. H. KLINE, B. S., AND E. B. ZIMMER, B S.,
CLEVELAND, OHIO

(New York Meeting, February, 1916)

It is well known that many properties of a given metal vary with the size of grain or cell. For most industrial purposes, where high ultimate strength and high elastic limit are desired, the manufacturer tries to produce a fine-grained structure. For some purposes, for example in transformer iron, a coarse-grained structure is desirable.

The terms fine-grained and coarse-grained are used only in a relative sense. For example, a fine-grained cast copper might have grains 100 times larger than coarse-grained high-speed steel. Similarly, a fine-grained steel rail might have grains or cells 100 times larger than coarse-grained wire, which was made of steel from the same heat.

In some metals, the change in grain size is more appreciable than the changes in any of the properties determined by the tensile test. Furthermore, in metals which have been annealed or subjected to high temperatures, grain size may prove to be a better indication of the life of the metal in use than the tensile test. One of the authors was able to supplant the tensile test with grain-size determinations in the control of metal for a specific use, the latter determination being more indicative of the life of the metal part than the tensile test. A grain-size determination with a tensile test is, of course, more valuable than either singly.

At present there is a decided dearth of data on actual grain-size determinations in metals, and consequently a corresponding lack of ability to interpret the determinations after they are made. It would be desirable to have correlation made between grain size and properties of metals, comparing thousands of determinations. In this way, grain-size determinations might be interpreted in much the same manner that tensile tests are now.

Several Methods of Measuring Grain Size

One of the chief reasons why more grain-size measurements have not been made is the excessive work involved in making a determination.

* A contribution from the metallurgical laboratory of Case School of Applied Science, Cleveland, Ohio.

† Instructor in Metallurgy, Case School of Applied Science.

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The planimeter method is accurate—in fact, too accurate for the nature of the work involved—but it is slow and tedious. It involves the tracing of the outside boundary lines of a group of grains, counting the grains, measuring the area with a planimeter, and a calculation which is different for each determination. Furthermore, the method requires a planimeter, which piece of apparatus might not be available in all metallographic laboratories.

We have developed a method by means of which a grain-size determination can be made in one-fifth the time required by the planimeter method. The method is as accurate as the sampling of the specimen and does not require apparatus which is not available in metallographic laboratories.

In this paper it is proposed: (1) to outline the more important methods now used for the determination of grain size in metals, (2) to outline a new method developed by us, and (3) to compare the various methods.

The Planimeter Method

This method may be used in either of two ways. The image of the metal specimen may be projected on to a piece of paper and the grain

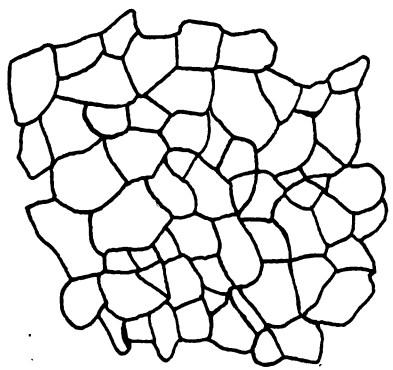

FIG. 1.—PLANIMETER METHOD.

boundaries traced, as shown in Fig. 1. The total area divided by the number of grains inclosed will give the average area per grain. The second way, which involves less time, is to trace on the paper just the outside lines bounding the area of a group of whole grains. The number of included grains is then indicated by check marks on the paper, as shown in Fig. 2. Since the counting of the grains can be done at the time of checking, this way is the shorter. The following example is taken from an actual determination:

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The area found by measuring with a planimeter was 5.92 sq. in. The number of grains inclosed was 61. Therefore, the average area per grain is $\frac{5.92}{61} = 0.09705$ sq. in. The observation was made at a magnification of 100 diameters. Then $\frac{0.09705}{100 \times 100} = 0.000009705$ sq. in., which is the actual area per grain.

$$\frac{1}{0.000009705} = 103{,}040 \text{ grains per square inch.}$$

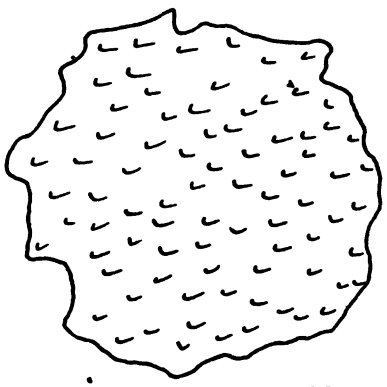

FIG. 2.—PLANIMETER METHOD, MODIFIED.

Heyn's Method

The Heyn method[1] depends for its accuracy on the assumption that the intercepts of a straight line intersecting a number of grains will be proportional to the square roots of the areas of the grains. In other words, the square of the average intercept will be the average area per grain. Heyn applies this assumption as shown in Fig. 3. The number of grains is counted along a line such as *AB*. The first incomplete grain is counted as one and the last grain is not counted. By taking a number of lines parallel to *AB* and counting the grains intersected by each, the average intercept in one direction is obtained. The average intercept in the direction *CD*, at right angles to *AB*, is obtained in a similar way. The product of the two average intercepts is the average area per grain. For example, suppose the number of intercepts in the direction *AB* is 79 and the distance 20 in. The average intercept is $\frac{20}{79} = 0.2533$ in. Likewise, suppose the number of intercepts counted in the direction *CD* is 72 and the distance is 16 in. The average inter-

[1] Reports of the Technische Hochschule zu Charlottenburg, *The Metallographist* vol. vi, pp. 54 to 63 (1903).

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cept in this direction is $\frac{16}{72} = 0.222$ in. Then the average area per grain is $0.222 \times 0.2533 = 0.05623$ sq. in. If the magnification is 100 diameters, then $\frac{0.05623}{100 \times 100} = 0.000005623$ sq. in., actual area per grain, or $\frac{1}{0.000005623} = 177{,}800$ grains per square inch.

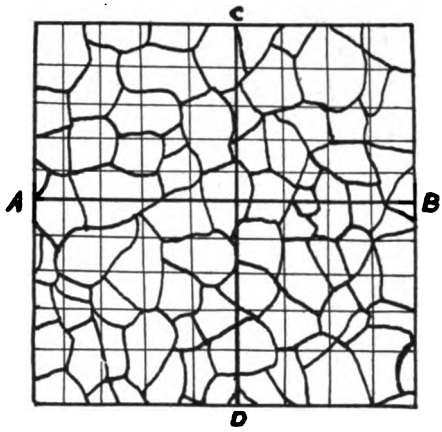

FIG. 3.—HEYN METHOD.

It is not necessary to trace the grain boundaries to apply Heyn's method. A piece of cross-section paper may be used for a screen and the grains intersecting each line counted and the distance measured.

The Intercept Method

The intercept method is based on the same assumption as Heyn's method, but the readings are not confined to series of parallel lines. The method of procedure for determining the grain size by the intercept method is as follows: In Fig. 4, the line AB is drawn through the center of the field. This line intersects 13 grains in a length of 2.58 in. The average intercept per grain is therefore $\frac{2.58}{13} = 0.198$ in. $(0.198)^2 = 0.0392$ sq. in., the average area per grain. If we consider the magnification as 100 diameters, then $\frac{0.0392}{100 \times 100} = 0.00000392$ sq. in., the actual area per grain. The number of grains per square inch is $\frac{1}{0.00000392} = 255{,}000$. In order to get the average diameter per grain, sufficient lines should be taken to intersect from 50 to 500 grains.

In actual practice, this method is shortened considerably.[2] An

[2] H. M. Howe: Life History of Network and Ferrite Grains in Carbon Steel, *Proceedings of the American Society for Testing Materials*, vol. xi, pp. 380 to 381 (1911).

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eye-piece with a graduated scale may be used in place of the regular eye-piece. The length of an even number of grains can be measured direct and the approximate grain size obtained in a short time.

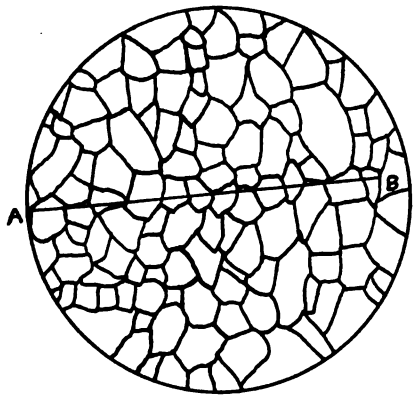

FIG. 4.—INTERCEPT METHOD

Authors' Method

Our method has for its object the quick determination of the equivalent number of whole grains included within a circular portion of an image. It is evident that some of the grains will be completely included within

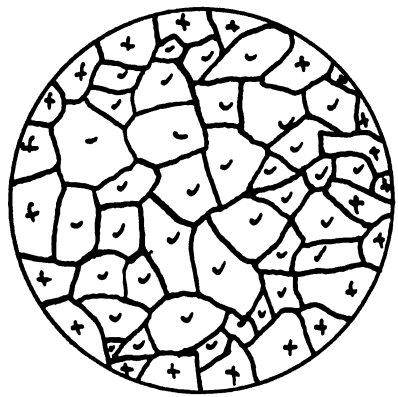

FIG. 5.—AUTHORS' METHOD.

and some will be intersected by the circumference of the circle. The former will have no correction factor. The boundary grains, however, will be partly inside and partly outside the circle. In Fig. 5, the grains marked with a check mark are completely inclosed and those marked + are the boundary grains. The average portion of each boundary grain included within the circle may be found as follows: The grains completely

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included within the circle are first counted and checked. Then, the boundary grains are checked with a different mark and counted. (If the counting is done at the time of checking, the time of making a determination will be shortened.) In Fig. 5, there are 37 included grains and 22 boundary grains. The area of the included grains by planimeter is 1.9 sq. in. The area of the circle (diameter 1.83 in.) is 2.635 sq. in.

$$1.9 : 2.635 = 37 : X$$

$X = 51.3$ the equivalent number of whole grains within the circle.

$51.3 - 37 = 14.3$ the equivalent number of whole grains in the inside portions of the 22 boundary grains. $\frac{14.3}{22} = 0.65$ average portion of each boundary grain within the circle.

If this factor be determined on a large number of samples, it may be used on unknown samples to find the equivalent number of whole grains within a circular area. If we call this factor y, the total number of grains within a circle is,

$$z + yw,$$

where z is the number of grains completely inclosed and w, the number of boundary grains.

The factor y has been determined empirically on 175 samples. The results are given in Table I. The first column represents the number of grains completely included within the circle.

TABLE I.

Grains Completely Included	Average Value of y	Number of Determinations
10–20	0.556	11
20–30	0.582	15
30–40	0.542	13
40–50	0.574	10
50–60	0.590	11
60–70	0.600	12
70–80	0.576	11
80–90	0.577	10
90–100	0.596	10
100–120	0.551	11
120–140	0.618	11
140–160	0.546	10
160–180	0.566	10
180–200	0.558	11
200–230	0.606	10
230–290	0.622	9

The average value of the factor y for 175 determinations is 0.581. The same factor can be used for all numbers of grains included within the observed area, but it is recommended that all important determinations have at least 50 grains included within the circle. Since 0.581 is nearer to 0.6 than to 0.5 we have adopted 0.6 as a general factor for all determinations. It should be noted that an error of 1 per cent. in the determination

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of the whole-grain equivalent of the boundary grains makes an error of less than 0.25 per cent. in the final result, since more than 75 per cent. of the total area of the circle is occupied by the included grains.

The following is an example of how the factor is used:

Inclosed grains = 66.0
Boundary grains = 28.0
Factor = 0.6

$66 + 0.6 \times 28 = 82.8$ grains within the circle.

From the magnification, the number of grains per unit of area can be readily calculated.

Comparison of the Various Methods with the Planimeter Method

In making the calculations for the comparison of methods, the actual number of grains per unit of area was not figured, the number of grains per square inch on the drawing being sufficient for comparison. The percentage of error was figured in the following manner. Suppose the planimeter method gave 50 grains per square inch, the intercept method 54.2, Heyn's method 55.6. The percentage of error in the intercept method is $\frac{4.2}{50} \times 100 = 8.4$ per cent.; and in Heyn's method $\frac{5.6}{50} \times 100 = 11.2$ per cent. Tables II, III, and IV show the comparative results obtained by using the different methods.

Table II.—*Heyn's Method*

Measurements in Direction *AB*			Measurements in Direction *CD*			Area of Grain		Number of Grains per Square Inch		
Total Length	Total Grains	Average Diameter	Total Length	Total Grains	Average Diameter	Heyn's Method	Planimeter Method	Heyn's Method	Planimeter Method	Error, Per Cent.
55.0	177	0.311	57.0	210	0.272	0.0845	0.0885	10.91	10.67	2.25
55.0	192	0.288	57.0	200	0.285	0.0820	0.0815	12.20	12.25	0.41
55.0	210	0.262	55.0	189	0.291	0.0761	0.0818	13.20	12.23	7.90
57.5	171	0.338	57.5	172	0.334	0.1130	0.1320	8.89	7.60	17.00
40.0	104	0.374	40.0	102	0.392	0.0151	0.1540	6.62	6.50	1.84
47.5	189	0.251	47.5	184	0.258	0.0650	0.0743	15.40	13.50	14.10
37.5	147	0.255	35.0	144	0.242	0.0630	0.0660	15.90	15.20	4.60
40.0	187	0.214	40.0	157	0.255	0.0512	0.0586	19.20	17.10	12.25
32.5	116	0.280	32.5	125	0.260	0.0730	0.0730	13.70	13.70	0.00
32.5	120	0.271	35.0	130	0.270	0.0700	0.0960	14.30	10.40	37.50
37.5	101	0.371	40.0	102	0.392	0.1455	0.1895	6.90	5.30	30.10
32.5	93	0.349	30.0	80	0.375	0.1310	0.1720	7.62	5.81	0.69
42.5	98	0.434	42.5	108	0.394	0.1710	0.1795	5.85	5.60	4.47
27.5	82	0.386	30.0	90	0.333	0.1120	0.1200	8.91	8.33	6.96
37.5	104	0.361	37.5	99	0.380	0.1360	0.1510	7.36	6.62	11.20
30.0	89	0.337	30.0	100	0.300	0.1010	0.1290	9.90	7.77	27.50
32.5	95	0.342	30.0	95	0.316	0.1080	0.1360	9.29	7.36	26.10
25.0	67	0.374	27.5	75	0.367	0.1370	0.1730	7.31	6.10	19.90
20.0	51	0.392	20.0	50	0.400	0.1570	0.1950	6.38	5.12	24.50
24.0	123	0.195	22.0	118	0.186	0.0365	0.0374	27.50	26.80	2.61
26.0	151	0.172	24.0	144	0.167	0.0287	0.0319	34.90	31.40	11.20
Average percentage of error										13.05

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TABLE III.—*Intercept Method*

Total Length of Line, Inches	Total Number of Grains Cut	Average Diameter of Grain	Area of Grain	Area of Grain by Planimeter	Planimeter Method, Grains per Square Inch	Intercept Method, Grains per Square Inch	Error, Per Cent.
16.70	87	0.1920	0.0369	0.0425	23.5	27.1	15.30
20.75	121	0.2275	0.0540	0.0353	28.4	19.5	31.30
17.80	98	0.1820	0.0330	0.0380	26.4	30.4	15.20
19.63	105	0.1780	0.0317	0.0412	24.3	31.5	28.90
18.50	107	0.1725	0.0297	0.0352	28.4	33.6	18.20
20.20	112	0.1810	0.0325	0.0357	28.0	30.8	10.00
20.45	115	0.1780	0.0317	0.0387	25.9	31.5	21.50
17.60	99	0.1780	0.0317	0.0438	22.9	31.5	37.50
20.30	119	0.1703	0.0291	0.0334	30.0	34.4	14.80
31.40	114	0.2760	0.0763	0.0875	11.4	13.1	14.90
34.70	135	0.2570	0.0660	0.0815	12.3	15.1	23.20
30.50	144	0.2120	0.0450	0.0529	19.0	22.2	17.10
32.52	120	0.2720	0.0740	0.0773	12.8	13.6	4.60
32.06	139	0.2310	0.0531	0.0810	12.4	18.8	51.50
33.58	135	0.2490	0.0620	0.0641	15.6	16.2	3.82
33.88	139	0.2450	0.0596	0.0682	14.6	16.2	15.00
33.88	138	0.2460	0.0605	0.0885	11.3	16.5	50.20
Average percentage of error.....							21.90

TABLE IV.—*Authors' Method*

Included Grains	Boundary Grains	Factor	Area of Circle	Authors' Method, Grains per Square Inch	Planimeter Method, Grains per Square Inch	Error, Per Cent.
118	49	0.6	7.068	147.4	146.30	0.8
95	35	0.6	7.068	116.0	123.80	6.3
34	27	0.6	7.068	50.2	49.85	0.7
59	30	0.6	7.068	77.0	76.22	1.0
74	37	0.6	7.068	96.2	94.93	1.3
56	35	0.6	7.068	77.0	73.69	4.5
110	41	0.6	7.068	134.6	137.80	2.3
93	40	0.6	7.068	117.0	116.97	0.0
75	37	0.6	7.068	97.2	94.20	3.2
103	43	0.6	7.068	128.8	134.00	3.9
Average percentage of error.....						2.4

In Heyn's method and the intercept method, the results are persistently high. In the former, all of the errors are plus except one, and that is minus only 0.41 per cent. In the intercept method, only one minus error is found.

In our method, the average of plus and minus errors is minus 0.1 per cent., which, of course, is less than the accuracy of the determinations.

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Quick Method of Grain-size Measurement by Authors' Method

A quick way to make a grain-size determination by our method has been developed. A circle 79.8 mm. in diameter is used. In making a grain-size determination, one of the magnifications given in Table V is used. In the third column is given the factor, for each magnification, by which the equivalent number of whole grains within the circle is to be multiplied to obtain the number of grains per square millimeter.

Table V.

Magnification Used	Diameter of Circle, Millimeters	Factor
25	79.8	¼
50	79.8	½
100	79.8	2.0
150	79.8	4.5
200	79.8	8.0
250	79.8	12.5
500	79.8	50.0
750	79.8	112.5
1,000	79.8	200.0
1,500	79.8	450.0
2,000	79.8	800.0

The following example shows how this table is used: Included grains, 61; boundary grains, 35; magnification, 100. Grains per square millimeter $= (61 + 0.6 \times 35)2 = 164$.

Similar tables can be made for square-inch calculations, or for circles of different diameters.

Discussion of Results

Our results show that the intercept method and Heyn's method are not accurate.

The largest error found in 187 determinations by our method was 6.4 per cent., the average error 2.1 per cent., and the average algebraic deviation from the planimeter measurements, plus 0.25 per cent.

An average of 30 representative determinations showed that 78.5 per cent. of the total area of the circle was occupied by the included grains. This portion is determined accurately by our method. The factor is used in determining only 21.5 per cent. of the number of grains. An error of 5 per cent. in estimating the boundary-grain portion makes an error of slightly over 1 per cent. in the final result.

A square or any other shape might be used instead of a circle. It seems probable, if a square or rectangle were used, that the factor would be 0.5 instead of 0.6, when the circle is used.

Suggested Rules for Sampling

Some metallographic specimens have more than 5,000,000 grains exposed on one face. It becomes necessary to obtain a representative

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sample of this face. We have tentatively adopted these general rules which, of course, can be changed as occasion demands.

1. Use such a magnification that about 50 grains will be included within the circle. (It is better to have more than 50 grains included within the circle than less, but it is always recommended that an even number of magnifications be used, even though the number of included grains should exceed 100.)

2. Make at least five determinations at about equal intervals along the diameter of the specimen. The average of all of the determinations is taken as the final result. It is evident that if there is much difference in grain size between the edge and center of the specimen, the former should have the greater weight. Such a case, however, may come under rule 4.

3. If the samples are small enough (for example, small wire), it is advisable to determine the total number of grains on transverse sections, and the total number for a given length on longitudinal sections. It is sometimes practical to do this with larger coarse-grained specimens either macroscopically or at low magnifications.

4. When one part of a specimen differs greatly from another part in grain size, do not average the separate determinations, but record each and note the position of each determination by means of a free-hand sketch.

5. In samples which have been cold worked, the relative degree of cold work in any direction is expressed by the ratio of the length divided by the width of the grains. For obtaining this ratio, it is recommended that Heyn's method be used.

Apparatus and Manipulation for Grain-Size Measurements

Any good metallurgical microscope with camera attachment can be used for grain-size determinations. It is recommended that only even magnifications be used. For instance, if an observation is being made at a magnification of 469 diameters, the setting should be changed to 500 diameters.

Grain-size measurements can be made on a screen or on a photograph. Either a ground glass, or a piece of paper fastened to a clear glass, may be used as a screen. A convenient system, where records of the determinations are kept, is to cut several pieces of thin typewriter paper so that they fit the clear-glass screen, and draw a circle with the desired diameter (we use 79.8 mm.) near the center of each sheet. Fasten one of these pieces of paper to the clear-glass screen by means of gummed labels, or otherwise, and project on to it the image of the specimen.

The circumference of the circle should be well within the image. Count and check separately the boundary grains and the included grains, make calculation for grains per unit area and save the sheet of paper

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with check marks and calculations. This will serve as a permanent record of the determination.

If the existence of a grain boundary is doubtful, often a slight change of focus will remove the doubt. It is, therefore, essential that the fine focusing screw be provided with an extension so the focus can be changed while the operator examines the image on the screen.

Unless the room is rather dark, or the microscope lamp excessively strong, the operator will find it helpful to work under a cloth hood.

The specimen should be etched so as to bring out distinctly the grain or cell boundaries.

Discussion

Zay Jeffries, Cleveland, Ohio.—The original portion of this paper represents a laboratory method for determining grain size in metals. It was developed not so much to find a quick method as it was to facilitate certain work which I was carrying on a couple of years ago. I had a great many determinations to make and the planimeter method was very slow, as was the Heyn method, and both of them were laborious, so I tried to shorten the time of the determinations.

Since preparing the present paper, Dr. T. M. Focke, Professor of Mathematics at Case School of Applied Science, has called my attention to the fact that both the Heyn and Intercept methods are mathematically incorrect, for the following reason: In any miscellaneous group of numbers the square of the mean is always less than the mean of the squares of those numbers. As an illustration, suppose we consider that the intercepts are in the ratio of 2, 3, 4, 5, and 6. The mean of these numbers is 4 and its square is 16; but the mean of the squares is 18. Considering the latter figure as the most likely to be correct, the square of the mean (16) would represent only 88.9 per cent. In the Heyn and Intercept methods, the square of the mean is used for calculation. This would make the indicated area per grain low and, hence, the number of grains per unit of area high.

George C. Stone, New York, N. Y.—I would ask Mr. Jeffries if there is a serious error, from the presence of a small number of grains that vary largely in size from the average? Do a few large grains or a few small grains make a serious difference?

Zay Jeffries.—Yes. It makes a serious difference in the various properties of the metal. It makes a difference in the working properties, and also the physical properties, and as far as results are concerned, if there is very much difference in the size of the individual grains, that fact should be noted when making the determination.

Alfred C. Lane, Tufts College, Mass.—I have done some grain-size work for the igneous rocks, and I can agree with what you said as to

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the difficulty in numerically estimating grain when you have a variation in size of the different grains.

I wanted to ask Mr. Jeffries one or two points. Why is it that that correction factor should not be an even 50 per cent., as one would offhand think it might be? Secondly, is not this intercept method practically the same as the Rosiwal method sometimes used for measuring the relative proportions of constituents in the rock?

I think I have found certain theoretical considerations for believing that the rate of cooling should be proportionate to the areas, which is of course what you were getting (though here the grain is developed in heating) and therefore an advantage. For a long time I had considerable doubt in my mind as to whether the linear dimensions, the area dimensions, or the volumes, were the most significant as regards grain size.

Of course, they are more or less in proportion to the first, second, and third power. I found there that while, theoretically, the rate of cooling is more or less closely associated with the areas, practically the first linear dimensions were more convenient to use, because, according to an approximate equation, the size of the grain, as measured by the linear dimensions, varied almost directly with the distances from the cooling margin if there was not superfusion or undercooling.

ZAY JEFFRIES.—In answer to the first question, of course, we can only conjecture. The figure $^{58}/_{100}$, I believe, was the average of all of our determinations for that factor, and we have found that factor to be correct empirically; but our idea of it is that the area inside of the circle would be 50 per cent. if the chord of the circle were used instead of the arc, and that the arc would include a greater percentage of the area.

As to the second question, the Intercept or Heyn method is identical with the method of determining areas by the straight-line method, that is, intercepts of a straight line. In regard to the mathematics of that method, I saw an article in *Economic Geology* in which that was discussed a couple of years ago, and the equations given were to the effect that the straight-line intercepts were proportional to the volumes and that the areas were proportional also to the volumes; but the determination of the grain size and the determination of the areas are really two different problems. The determination of the relative areas of different constituents, on a certain section, involves either the planimeter measurement of those various areas or some method of estimation.

A method used in metallography for estimating the relative areas is one known as the ruled-square method, in which a cross-sectioned paper is used for a screen, and the number of magnifications is varied at will. The areas of one constituent are made to cover from one to several squares. The total number of whole squares occupied by that constituent is counted, and then the fractional squares are estimated. The number of squares occupied by that constituent divided by the total

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number of squares, used for a sample, represents the proportional area of that constituent.

C. H. FULTON, Cleveland, Ohio.—With regard to Dr. Lane's remarks on the percentage of error, he probably has in mind rock sections, while Mr. Jeffries is speaking of metal sections.

The crystallization in metals is ordinarily quite fine, so that the number of grains within a given area is large, while, if I remember correctly, the number of crystals in a rock section is relatively small, so that the percentage of error naturally would be much less in metal sections than in rock sections.

ALFRED C. LANE.—That is right; if I came within 10 per cent., I thought I was doing well.

W. M. CORSE, Niagara Falls, N. Y.—There are two practical questions that come up in my mind, after hearing the paper this morning. One bears on the corrosion of metals, and the other upon the wear of metals. Both of the results are interesting, from an engineering standpoint, and, as I understand this paper, the grain size, or the determination of grain size of metals, might throw some light on the corrosion of a certain metal or an alloy and also on its wear in such a piece as a bearing or a gear.

I should like to inquire if the results of the determinations published in the paper would indicate that any light could be thrown on the corrosion or wear of any particular metal under examination?

ZAY JEFFRIES.—Of course, the subject of corrosion is a large one and there is relation between grain size and corrosion; but there is another relation which is more marked than the grain size and that is the previous treatment of the metal, *i.e.*, whether annealed, or whether strained, the latter including, of course, cold work.

Now, in view of the amorphous cement theory, the fine-grained metal would have more amorphous material in it, and an amorphous material is conceded to corrode faster than crystalline material. For that reason the fine-grained metal should normally corrode faster than the coarse-grained metal.

As to whether that is true or not, I am not prepared to say, but that should be true from a theoretical standpoint. As regards cold-worked metals, there is absolutely no question. The cold working of a metal produces more amorphous material, both between the grain boundaries and inside the grains themselves, and the increased quantity of the amorphous phase is considered to be partly the cause of the more rapid rate of corrosion of cold-worked metals.

As a general rule, if the rapid tests for corrosion (such as solution tests) are an indication, I should say that a fine-grained material would corrode faster than a coarse-grained material. I know the volatilization

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of the amorphous material is greater than crystalline material, and a fine-grained material will lose more weight than coarser material, by volatilization alone.

As regards the wear proposition, that brings in the subject of hardness and it brings in the subject of possibly two or three kinds of hardness; resistance to abrasion, and resistance to deformation, for instance. While some materials excel in the resistance to abrasion, others will excel in the resistance to deformation.

The fine-grained materials are harder than the coarse-grained materials of the same chemical analysis. Considering them both in the annealed state, or in the strained state with a given amount of cold work, we find the fine-grained material to be harder than the coarse-grained metal, and it will be more difficult to wear a fine-grained material of a certain definite composition, than if it were coarse grained.

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Recrystallization of Cold-Worked Alpha Brass on Annealing*

BY C. H. MATHEWSON,† PHD., NEW HAVEN, CONN., AND ARTHUR PHILLIPS,‡ M. S., BRIDGEPORT, CONN.

(New York Meeting, February, 1916)

DURING the past year considerable work dealing with the mechanical properties and microstructure following the anneal under uniform conditions of certain types of commercial rolled brass has been done in this laboratory. Since these experiments were conducted with the active coöperation of the Bridgeport Brass Co., involving free use of its product, and have thus furnished information which is characteristic of some of its special mixtures, full publication of the numerical relationships and comparative results encountered does not seem justifiable. Some of the more general features of this work are, however, available and it is our present purpose to use the material in question, along with some results of special tests, in discussing the characteristics of recrystallization as related to the degree of hardening by strain and the ordinary annealing variables. We desire to make acknowledgment of the many courtesies extended by Mr. Charles Ferry, Metallurgist of the Bridgeport Brass Co., and the value of his coöperation, which has greatly stimulated the study of brass in this laboratory.

A clear and systematic presentation of the principal relations between temperature and time of anneal, the changing physical properties and microstructure and the degree of previous reduction by cold-rolling, was first made, in the case of rolled brass and copper, by the French engineer, C. Grard,[1] in 1909, although much fragmentary information of this general character had appeared earlier and the main facts involved were undoubtedly known to many brass specialists through their own private research. From this work, it appears that in the cartridge mixture (67 per cent. copper, 33 per cent. zinc) the properties imparted by cold-working within the limits, 13 to 75 per cent. reduction of area by rolling $\left(\frac{\text{Original Area} - \text{Final Area}}{\text{Original Area}} = 0.13 \text{ to } 0.75\right)$, are not affected by annealing for a 50-min. period at any temperature below 200°C. At or in the

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[1] C. Grard: Cuivre et Laitons à Cartouches, *Revue d'Artillerie*, Feb.–Apr., 1909.

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vicinity of this temperature, metal which had been reduced 75 per cent. showed a sensible decrease in tensile strength and a corresponding increase in elongation, while metal which had been reduced 13 per cent. showed no change. An intermediate value of reduction, 56 per cent., the one which Grard evidently used in most of his tests, gave earliest evidences of change at about 250°. Thus, the effect of anneal in its incipient stages depends somewhat upon the previous treatment of the metal. As the temperature of anneal is increased, the change in properties becomes more marked in each case, until, at about 325°, the individual curves coincide, and from this point, the final condition produced by anneal is independent of the previous reduction within the limits investigated. With regard to peculiarities of inflexion, Grard distinguishes a number of zones corresponding to regions of permanent, or of changing properties in one sense or another. In accepting such conclusions, the empirical character of the work must be borne in mind. Grard has shown, as indicated above, that the degree of reduction by rolling determines the temperature range of permanency in the original strain-hardened condition. The reasons for this and the prevailing possibilities of variation will at least be indicated from a general point of view in the present paper.

In the principal set of curves, Grard adopted a period of 30 min. in which to complete the anneal at any given temperature. From his temperature-time curves it appears that in all cases the rate of change of physical properties, which is greatest in the earlier stages of the anneal, has become very slow at the expiration of the whole period. Moreover, as the temperature of anneal is raised, a decreasing fraction of the total 30-min. period suffices to produce most of the annealing effect. That this is a generally prevalent condition was shown by T. K. Rose[2] who, in studying the annealing properties of a variety of coinage alloys by the scleroscopic method, makes the following statement: "The annealing action at any given temperature is most rapid at first and gradually dies away, so that a state of equilibrium is not attained in any short space of time, except perhaps at high temperatures. For example, equilibrium apparently was not attained in the gold-copper, silver-copper, or nickel-copper alloys, or in pure nickel, in 16 days at 300°. At high temperatures, almost the whole result is obtained instantaneously, though a little extra softness is got by prolonging the time. A few minutes at such a temperature cannot be replaced, for practical purposes, even by hours of annealing at much lower temperatures."

This work of Grard has been widely circulated among brass metallurgists through the medium of reprint, translation, and review. Doubtless it has supplied information of direct numerical value in some quarters,

[2] T. K. Rose: On the Annealing of Coinage Alloys, *Journal of the Institute of Metals*, vol. viii, pp. 86–125 (1912).

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while in others it has served as a welcome guide in the development and coördination of testing data. The latter type of service appears to us to be the most desirable function of publications of this sort. The individual brass mill may be urged to develop its own data of standardization, or control, whereby all local matters of manufacturing practice, peculiarities or specialties in composition, and other individual features, may receive consideration in their proper environment, leaving broader conceptions subject to development and modification by exchange of ideas through the scientific press.

With these ideas in mind, the following notes are selected from work which we have done, with the hope that they may be of some service to those who are interested in the annealing characteristics of brass. In these notes some addition to the commonly available information on brass metallurgy has been made chiefly along the following lines:

(*a*) By presenting a few carefully conducted physical tests to show the general direction and magnitude of changes which are effected by heating cold-rolled strips below an effective annealing temperature.

(*b*) By determining the periods of time necessary to produce a measurable amount of softening in one kind of strain-hardened material at a succession of temperatures below the region of rapid effects.

(*c*) By emphasizing the micrographic aspects of strain-hardening and recrystallization in metal which has received both light and heavy reductions by rolling.

(*d*) By discussing the relations between temperature and time of anneal, degree of deformation, and structural alteration in alpha brass, with an attempt to give some general explanation of the facts involved.

(*e*) By showing comparisons between the ordinary physical properties and the grain size, both given as functions of the annealing temperature for a fixed period of anneal.

Changes Produced in Cold-Rolled Strips by Heating One-Half Hour at 200°C.

Bengough's[3] experiments on 70/30 brass, while devoted largely to the question of "burning," extend over a wide enough range to furnish a broad basis of comparison with those of Grard. Both observers have recorded a similar sequence of changes in the strength-properties after a constant period of anneal at different temperatures, except that, according to Bengough, the metal becomes several per cent. stronger after anneal at temperatures in the vicinity of 300°, while Grard, in his curves, gives no indication of an increase in tensile strength before the normal (reverse) effect, due to anneal at higher temperatures, sets in.

It may be urged that variations of this order may be expected, in

[3] Bengough and Hudson: Heat Treatment of Brass, *Journal of the Institute of Metals*, vol. iv, pp. 92–127 (1910).

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view of unavoidable slight imperfections in the test pieces, or on account of testing errors, irrespective of any actual annealing effect. It is obviously necessary to obtain the mean values from a moderately large number of tests before reasonably certain conclusions can be drawn. We are not certain whether Bengough used several test pieces for each experiment, or relied upon a single piece to represent each anneal. In any event, it is true that, in all three series embracing low-temperature anneals, viz., copper 69.4 per cent., ¾-in. round bar; copper 71 per cent., ¾-in. round bar, and copper 71 per cent., ⅛-in. wire, somewhat higher tensile strength was obtained in samples annealed between 285° and 320° (different series), than in the original hard-rolled material. A critical examination of Grard's data is not possible, since the actual values are not tabulated or shown in his diagrams.

Our method of testing the point at issue was as follows: Strips of the dimensions, 0.1 by 0.5 by 8 in., were sawed from a cold-rolled sheet of 70/30 brass* which had received a dead soft anneal in the mill previous to the final reduction of 40 per cent. Five of these strips were tested in a 50,000-lb. Riehle universal testing machine in the cold-rolled condition. Another set of five was tested after annealing for a period of one-half hour at 200°. The material under present consideration cannot be heated for a half-hour period at temperatures in the neighborhood of 300°, viz., at temperatures similar to those given by Bengough, without appreciable softening and visible recrystallization (true annealing effect). One of our tests at 300° showed a softening of four points on the scleroscope scale and abundant recrystallization, as illustrated in Fig. A, Plate I. The main reason for this difference lies in the moderate initial reductions by cold-working received by Bengough's rods, as compared with the heavy initial reduction received by our strips. While Bengough does not give the reductions used in his material, the above conclusion is justified by the materially lower tensile strength and higher elongation of his bars, as compared with our strips; all tested in the cold-rolled condition.

The annealing was conducted in an electric resistance furnace of rectangular box-shaped type containing a central tube of alundum (14-in. long by 1 in. bore) closely wound with "Nichrome" ribbon cemented into place with alundum cement. To avoid rapid radiation from this heating unit, it was embedded in a mixture of granular fire clay and silica.

Two test strips were treated simultaneously (except in the annealing of the fifth strip). The strips were clamped in a vise and a hole large enough to accommodate the thermo-couple tube was drilled at one end in such manner as to bore both pieces simultaneously for a distance of approximately ½ in. in the direction of their length. Two wire clasps held the strips in this position and served as a support for the metal when placed

* Analysis: Copper, 69.35; iron, 0.04; lead, 0.02 per cent.

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in the center of the furnace tube. All bars for tests and micrographic observations were annealed in this way. After anneal, the drilled ends were cut off and micrographic observations made in the immediate vicinity of the region where the thermo-couple junction had previously rested. It is to be noted that the junction was virtually imbedded in the metal at this point.

Owing to increased radiation at the ends of the furnace, the temperature of the tube at the center was found by preliminary experiments to be about 10° higher than at the ends. This difference is small and is actually less in the bars themselves, as their high heat conductivity tends toward equalization of temperature, *i.e.*, flattens the thermal gradient.

Probably the most effective method of flattening the thermal gradient in a wire-wound tube furnace is to make use of independent heating coils around the ends which may be run at a temperature just high enough to compensate for the increased radiation at the ends. Gray[4] has obtained ideal conditions with this type of construction. By decreasing the distance between turns from center to ends, a compensating effect may be secured, but with any given winding the desired effect will be secured only when the furnace is running at certain temperatures; above and below this range, the ends will be over- and under-compensated, respectively.

Gray's method complicates the construction of the furnace and requires independent circuits along with independent control, while the other method requires a number of different resistors to be used for different temperature ranges and there is great difficulty in properly spacing the windings. We have sought to improve the gradient without introducing another element of control and without greatly complicating the construction of the furnace. Lately, we have used furnaces in which the thickness of the heat-insulating covering around the tube (granular fire-clay) increases from the center to the ends, thus permitting more rapid loss of heat in the region of the center. In principle, this is similar to varying the space between turns, but it is far easier to accomplish, and any adjustment may be changed in a few minutes. This modification merely requires the construction of false inner sides of sheet iron which are bent to give the proper inclination from center to ends and slipped within the furnace box (made of asbestos wood). The granular fire-clay is then filled in with a scoop. It is clear that, according to this construction, the thickness of the heat-insulating covering varies from the center of the tube to the ends, but the depth is constant, viz., the correction applies to the sides of the tube but not to the top and bottom. While it would be preferable to make the correction concentric with the tube, this would entail greater complication and less flexibility. These

[4] A. W. Gray: Production of Temperature Uniformity in an Electric Furnace, *Bulletin Bureau of Standards*, vol. x, pp. 451–473 (1914)

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furnaces may be taken down in a few minutes. They are capable of reducing the maximum variation of temperature along 8 in. of the central portion of the 18-in. tube from 30°C. to less than 10°C. within the range 200° to 1,000°, using a single adjustment.

In all of the annealing work, the test pieces were placed in a hot furnace and rapidly brought to the annealing temperature which was then kept constant for the desired period by adjusting the external resistance. With the present set, a preheating period of 10 min. is to be added to the annealing period of 30 min. The results are shown in Table I.

TABLE I.—*Early Effects of Annealing upon Physical Properties of Worked Brass*

No.	Treatment: Time and Temperature of Anneal	Tensile Strength in Pounds per Square Inch	Percentage Elongation in 2 in.	Hardness by Scleroscope
1	As rolled (40 per cent. reduction).....	76,829	10.0	33.2
2	As rolled (40 per cent. reduction).....	76,424	11.0	34.4
3	As rolled (40 per cent. reduction).....	76,822	11.0	33.0
4	As rolled (40 per cent. reduction).....	77,017	11.0	33.0
5	As rolled (40 per cent. reduction).....	76,723	12.5	33.2
Average	.....	76,763	11.1	33.2
6	½ hr. 200°C.....	78,415	10.5	34.0
7	½ hr. 200°C.....	79,166	10.0	34.2
8	½ hr. 200°C.....	78,615	10.5	34.4
9	½ hr. 200°C.....	77,914	9.0	34.0
10	½ hr. 200°C.....	78,450	10.0	34.2
Average	.....	78,512	10.0	34.2

The two sets of numerical values are consistent and seem to indicate that there is a slight but unmistakable increase in tensile strength and a corresponding decrease in elongation produced by heating at 200°C. It would seem that Bengough and Hudson's curves indicate more accurately than Grard's the true conditions brought about by heating at low temperatures.

The precise nature of the changes which take place in worked metal as a result of exposure to very moderate temperatures, or even spontaneously, is not understood. Any attempt to study these changes with the aid of the microscope is certain to fail, since the accompanying structural alterations are too minute or too indefinite to be followed. In the present case, careful comparisons between the two sets of strips were made with the aid of a $\frac{1}{12}$-in. immersion objective without detecting any difference. Both the heated and the original strips show the typical folds, or wrinkles, which are prominent in heavily worked brass and may be seen under low power in the photomicrographs of Plate II, or under high power (× 1,000) in Fig. A, Plate I, leaving out the central, recrystallized portion.

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It is obvious that since no alteration of structure can be associated with these effects they must be of a mechanical nature, or they must be due to some reorganization of the particles which does not suffice to change the etching characteristics of the metal. The former interpretation is practically substantiated by the recent work of E. Heyn[5] in which the distribution of internal strains in cold-worked metal before and after heat treatment is determined by means of special tests. This subject is handled at length and in a masterly manner by Heyn. We will merely draw attention to his observation that reheating at temperatures even below the softening point is productive of "remarkable effects toward diminishing the degree of internal strains" with the comment that similar effects in the present experiments are in all probability responsible for the differences observed in the ordinary tensile test. Heating at or above the earliest softening point would entirely remove internal strain but would also lower the tensile strength. It would seem from the experiments in hand that the frequent destructive effect of aging (season-cracking) in severely worked metal, which is undoubtedly due to relief of internal strain from some local cause, could be avoided in every case by suitable heat treatment without loss of tensile strength. The present tests show that there may actually be an increase in tensile strength associated with the partial release of strains in some rolled shapes.'

Time Required at a Number of Different Temperatures between 225° and 325°C. to Produce a Drop of 3 Points in Scleroscopic Hardness in the Case of 70/30 Brass which had Received 40 Per Cent. Reduction by Rolling after a Dead Soft Anneal in the Mill

The same material was used in these tests as in those just described. A number of 1-in. pieces cut from the 8-in. strips were placed in the furnace at the given temperature and removed at intervals.

TABLE II.—*Factors Corresponding to a Drop of 3 Points in Scleroscopic Hardness after 40 Per Cent. Reduction by Cold-Rolling*

Temperature of Anneal	Time of Anneal
°C.	Hours
225	294
240	189
255	46
275	5¼
300	2½
	Minutes
325	18

[5] E. Heyn: Internal Strains in Cold-Wrought Metals and Some Troubles Caused Thereby, *Journal of the Institute of Metals*, vol. xii, pp. 3–37 (1914).

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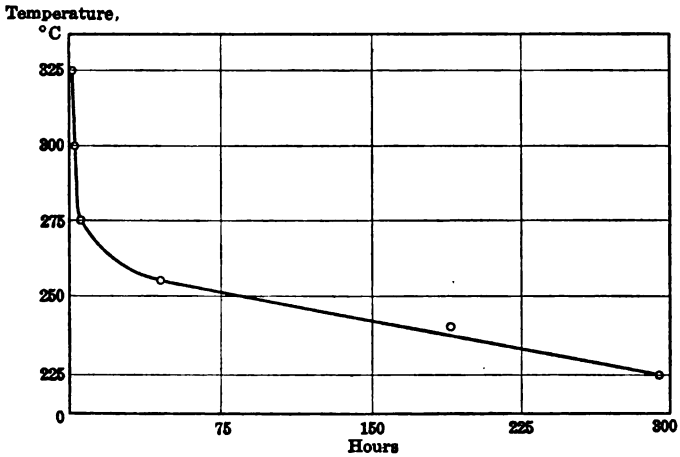
At 425° and 375°, the strip pieces could not be brought to the furnace temperature, which required about 5 min., without losing more than the desired 3 points on the scleroscope scale. In the former case, the loss was 13 points, and in the latter, 8 points. Between 325° and 225°, the time necessary for a 3-point drop could be measured and the values obtained are given in Table II.

Fig. 1 shows the above results in graphical form. From a practical standpoint, if we are to speak of a "critical" temperature at which softening will begin in any particular mixture which has been given a stated reduction by cold-working, we must not omit certain qualifications, since the same effect will be produced at a number of different temperatures depending upon the time allowed. From the indications of this curve and from other considerations to be shown later, there is reason to believe



Time necessary to cause drop of 3 points in scleroscopic hardness.

FIG. 1.—CARTRIDGE BRASS (70/30) REDUCED 40 PER CENT. IN AREA OF SECTION BY COLD-ROLLING.

that there is a minimum temperature corresponding to a given degree of reduction at which the first softening effects may occur, but at this temperature the change proceeds very slowly, so that a period of days, or months, may be required to bring about equilibrium. The particular equilibrium involved will be discussed in another section of this paper.

In the present case, the arbitrarily chosen unit of measurable softening* is realized at a temperature as low as 225°, but only after a period of about 12½ days, while, at a temperature only 50° higher, the same effect is produced in about 5 hr., and at a temperature still higher by 50°,

* We have chosen 3 points on the scleroscope scale as the measure of appreciable softening since there can be no doubt from an experimental standpoint that the metal has softened when this indication is obtained. A drop of 2 points is not as easy to detect, while an indicated drop of 1 point is always open to doubt.

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viz., at 325°, the effect is produced in a period of minutes. None of the values shown on the curve are equilibrium values, since further softening occurs at any given temperature on longer exposure; from the flattening of the curve, however, it is apparent that the lowest value given, viz., 225°, is not far above the true equilibrium temperature for this degree of softening.

Bearing these facts in mind, it is clear that any set of curves giving the physical properties of a commercial mixture after anneal for a fixed period of time following a given reduction by cold-working requires that the stated period of anneal be rigidly adhered to within the critical range of incipient effects, in order that the results of test may be properly duplicated. At higher temperatures, when all changes are rapid, approximate equilibrium values are reached in comparatively brief periods of time and some latitude in this respect may be allowed without materially affecting the results. For example, in the case of cartridge brass (70/30±3), which has received a heavy reduction, at all temperatures above 325°C., variation of the annealing period between ½ hr. and 1 hr. will not bring about material change in properties; at somewhat lower temperatures, however, incipient effects are encountered and the time factor must be carefully established by test.

Early Stages of Recrystallization in Material which has Received a High Degree of Reduction by Cold Work

Rose observes[2] that the change in hardness and the recrystallization, in the case of gold, take place almost simultaneously, with a slight lag in the visible recrystallization; especially if the specimen is heated for a short time only. As far as can be inferred, his observations were all made at low magnification and, in case of a slight drop in hardness, he failed to detect recrystallization. As a result of the experiments just described, we are in possession of a large number of small pieces of metal which show different degrees of hardness varying from 33, that of the original cold-rolled metal, to 14, and these variations were brought about by heating for different periods of time at the various temperatures within the range, 225° to 325°. A sufficient number of these were carefully examined to prove beyond doubt that any positive indication of softening by the scleroscope can be detected in the form of recrystallization under the microscope. This applies only to material which has received comparatively heavy reductions, as will be pointed out more clearly in a later section.

In order to detect recrystallization of this order, microscopic examination under high-power is necessary, as the first visible elements of recrystallization are extremely minute. Before describing these effects,

[2] *Journal of the Institute of Metals*, vol. viii, pp. 86–125 (1912).

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it may be well to refer briefly to the ordinary appearance of brass after different degrees of strain-hardening, as seen under the moderate magnifications usually adopted in brass work. A set of photomicrographs* representing the effect of cold-rolling in stages from gages 18 to 25 (B. & S.) using alpha brass containing about 64 per cent. copper, 0.2 per cent. lead and 0.04 per cent. iron is shown in Plate II. The reductions are given with the micrographs. The samples were etched with ammonia and hydrogen peroxide. The common high brass, from which these micrographs were made, and cartridge metal, which is the subject of the present experiments, both show the same deformational characteristics. It may be noted that Fig. F, Plate II, represents approximately the same degree of reduction as that received by the metal used for the present tests.

In general, reductions of 5 or 10 per cent. have little or no effect on the etching properties of alpha brass. Photographs taken before and after deformation are very similar; practically the only alteration due to moderate working of the metal which may be observed in the photograph is a slight displacement of the normally rectilinear boundaries between parts of a twinned grain, so that they frequently appear curved. Some of these curved bands may be seen in Fig. B, but, more prominently, in the succeeding figures. When the reduction reaches about 20 per cent., the etching characteristics of the metal begin to change; there is a loss of contrast between differently orientated grains owing to the development of a secondary structure within the grains themselves. This is barely discernible in Fig. C, but becomes more prominent as we pass to higher reductions in the succeeding figures. At a reduction of about 30 per cent., as in Fig. D, the direction of extension in cold-working becomes apparent, in that the grains themselves show a prevailing direction of elongation.

The intergranular secondary structure due to deformation consists in the main of groups of curved or wavy lines which possess the same general direction in each grain, or homogeneous portion of a twinned grain, and increase in number and prominence with the degree of reduction. They incline to a direction at right angles to the direction of elongation and it appears that, in the squeezing of the grains between the rolls, the most destructive movements affecting the integrity and homogeneity of the grain substance have occurred chiefly along the gliding planes which, in any given grain, are nearest at right angles to the direction of elongation.

In the absence of any generally accepted term of definition, these lines may be called, simply, lines of deformation. They are the most prominent and the most characteristic lines which are produced by any deformational process. In some laboratories, the term slip bands has been dis-

* These photomicrographs were prepared by Philip Davidson in the Hammond Laboratory for the Scovill Manufacturing Co. who have kindly permitted us to use them in the present connection.

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torted out of its original meaning and made to include markings of this character. It seems wiser to restrict this term to its original meaning which relates only to the visible effects produced when associated units of grain-substance move along the crystallographic gliding planes and break the plane of the polished surface. It does not appear wholly correct to make use of the term, Neumann's lines, in this connection, since these markings, which are revealed on etching and doubtless represent etching characteristics dependent upon a previous form of mechanical treatment, have been studied chiefly in connection with meteoric iron, in which they were originally discovered (Neumann, 1850), and have received an authoritative interpretation based upon crystallographic relationships in the case of iron alone.[6, 7]

When a specimen of rolled brass showing lines of deformation is manipulated under the higher magnifications, it is seen that the black lines are shade effects primarily due to differential etching of the surface into curved elements of varying breadth and elevation. The surfaces of intersection of neighboring elements reflect the incident rays away from the line of vision producing dark bands whose breadth and quality vary with the focussing position and the angle of illumination. Where the original orientation of the grain is such as to reflect very little light to the eye, *i.e.*, in the case of a dark grain, the deformational detail may yet be seen to some extent, since part of the light reflected from the curved surfaces of intersection between bands reaches the eye. Thus, the light grains as a whole are rendered darker, while the dark grains are rendered lighter, so that the original contrasts are greatly reduced. This is apparent in the series of photomicrographs shown in Plate II.

The deformational detail of several grains may be seen in Fig. A, Plate I, under a magnification of 1,000. The recrystallized central portion of the figure need not be considered in the present connection. An acute-angled grain in the upper right-hand portion of the photograph shows a uniformly close arrangement of bands, wrinkles, or folds, as they may be variously termed. It may not be inappropriate to pattern after the term slip bands, and designate these bands, which constitute the etching record of intercrystalline deformation, *etch bands*.

The grain shown in the lower left-hand portion of the figure is marked with wider bands, while the grain just above this has reflected little light upon the plate and gives an unsatisfactory rendering of detail. In some cases, the etch bands are crossed by a second set of bands; this is seen in the extreme upper center of the photograph.

The edges of the etch bands are frequently serrated, as shown in Fig. B, Plate I. This indicates that the existing positions have been reached

[6] Osmond, Fremont and Cartaud: Les Modes de Déformation, *Revue de Métallurgie*, i, p. 11 (1904).

[7] Osmond and Cartaud: The Crystallography of Iron, *Journal of the Iron and Steel Institute*, iii, p. 444 (1906).

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through movements of step-form, which is in accord with present conceptions of plastic deformation. It is probable that serrated edges would always be observed if adequate magnifications could be used.

While direct observations of this character cannot fail to aid the imagination in devising theories to account for the series of transformations which are effected by severe mechanical treatment followed by annealing the actual processes of reorganization cannot be observed owing to the minute character of the participating units. A few remarks bearing upon this subject will be made in following paragraphs, along with brief reference to leading theories.

In a recrystallized structure, free from mechanical alteration, each homogeneous member or grain shows its individuality on etching by reason of its directional properties, which determine selective etching with boundary distinctions, or selective reflection with contrast effects. When the grains are strained beyond the limit of elasticity, their temperature range of existence without visible change of form is restricted. On heating to a high enough temperature, they appear to disintegrate into smaller grains. It is, however, entirely at variance with the laws of energetics for large grains to disintegrate spontaneously into smaller grains and thereby accumulate additional surface energy. Therefore, the increase in surface energy must have developed as a result of the strain, *i.e.*, in the form of inner surfaces between blocks, or groups, of the grain substance. This mechanical dislocation of the grain substance is not indicated in any way by etching except in cases of severe strain. For example, the inner surfaces which must exist after a 5 or 10 per cent. reduction by rolling, since the metal will thereafter develop a secondary structure on annealing, are not indicated in the corresponding photographs, Figs. A and B, respectively, of Plate II. A more complete illustration may be had by referring to Plates VII and VIII, the first of which represents cartridge metal which has received an 8 per cent. reduction, followed by a series of annealed structures; and the second, similar metal which has received a reduction of 12 per cent. followed by a series of annealed structures. In neither case has the rolling produced any clearly discernible dislocation within the individual grains, but, in both cases, the coarse grain structure is refined by annealing at 550°.

It is thus evident that we cannot see the inner surfaces and we can only speculate as to their nature. Since differences in orientation are ordinarily revealed on etching, it may be argued that the deformation has not brought about differences in orientation, but has merely displaced blocks of component particles along the natural gliding planes. If this is all that has occurred, it is difficult to see why any reorganization should result on annealing, since the particles are still arranged in conformity with normal crystallographic requirements, *i.e.*, the original space-lattices are preserved.

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As a matter of fact, a sample of brass which had been heated to 750° so as to develop a coarse grain and then strained so as to show numerous slip bands on the polished surface (see Fig. A, Plate III), gave little or no evidence of recrystallization after annealing at approximately 600° (see Fig. B, Plate III). The sample was photographed in the original position after annealing and, making due allowance for changes in the dimensions of the grains as a result of repolishing (in which several thousandths of an inch was removed from the surface), it is clear that the original structural outlines are generally preserved in the annealed metal.

Tammann's theory of plastic deformation[8] is founded upon crystallographic principles and admits only of movements of translation, or of movements into twinning position, along the natural gliding planes. The occurrence of inner surfaces along the gliding planes would necessitate their subsequent removal on annealing, but no reason is apparent why this should not be effected by recoalescence in normal alignment with preservation of the original space-lattices. There would then be no recrystallization and the grain would acquire stability in its altered form.

It is not impossible that slight differences in orientation may occur without influencing the etching characteristics to a visible extent. What appears to be a single grain may, in some cases, be a close association of two grains which chance to have about the same orientation. We have frequently observed cases where the distinction between two neighboring grains was difficult to make. Thus, the deformational process may bring about surfaces of discontinuity between blocks of material within a single grain, whereby the orientation changes only slightly from block to block. There would then be opportunity for recrystallization on anneal with accentuation of visible differences of orientation, as the normal process of coalescence brings about a survival of those grains, or reorganized blocks of material, which possess the strongest individualistic tendencies and can maintain contact existence.

Plastic deformation, according to Lehmann[9] (see also Czochralski[10]), results in an altered arrangement of the grain substance under the combined influence of the imposed forces and those forces which ordinarily operate within the molecular aggregate. Thus, blocks or groups of particles are forced into a new position of more or less orderly arrangement in which certain axial and other relationships are assumed. A conception of this sort would imply far-reaching reorganization of the grain substance at elevated temperature, since forms of natural stability

[8] G. Tamann: *Zeitschrift für Elektrochemie*, vol. xviii, p. 584 (1912); also *Lehrbuch der Metallographie* (Leipzig, 1914), Chapter 1, C.

[9] O. Lehmann: Spontane und Erzwungene Homoötropie, *Internationale Zeitschrift für Metallographie*, vol. vi, pp. 217–237 (1914).

[10] Czochralski: Gegen die Translations-Hypothese, *Internationale Zeitschrift für Metallographie*, vol. vi, pp. 289–296 (1914).

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(recrystallized units) would then replace the mechanically organized forms, with removal of artificial inner surfaces.

It is but a step from the above theory of displacement to the amorphous theory developed by Rosenhain[11] as an extension of Beilby's earlier work in this field. Instead of an orderly displacement of the affected particles, this theory would lead us to believe that truly amorphous membranes, or envelopes, are formed along the articulating surfaces. In other words, certain of the grain particles are rubbed out of alignment and given an isotropic arrangement when blocks or layers of grain-substance move against one another. Hardness and strength are supposed to be specific properties of the amorphous cementing material, thus explaining strain-hardening at ordinary temperature. The inner surfaces, in this case, are the surfaces of contact between the two molecular varieties of material and the disappearance of such surfaces at elevated temperature, owing to transformation of the unstable amorphous material into stable crystalline material with reorganization and growth of grain, is a rational development.

No adequate discussion of the merits of these theories is contemplated in these pages; it has merely seemed advisable to have the main points of the most prominent theories in mind while considering such visible effects of structural reorganization as we have been able to define in the case of brass. As previously pointed out, we cannot hope to see the early stages of this reorganization under the microscope for obvious reasons.

Turning again to the photomicrographs of rolled metal, it may be remarked that inner surfaces of the nature required by Rosenhain's theory would, in all probability, fail to show on etching owing to the extreme thinness of the amorphous membranes which Rosenhain conceives to be of molecular dimensions. When the deformation reaches a high value, as shown in the last five figures of Plate II under low power, and in the first figure of Plate I under high power, the etch bands appear. This constitutes direct evidence that the material within a given grain is broken up into groups which differ among themselves in some fundamental essential. The mere fact that there are etching distinctions indicates that there are also underlying distinctions affecting the grouping and alignment of the component particles. When these groups reorganize on annealing, the banded form is entirely lost; in other words, a band of uniform appearance under the microscope disintegrates and participates in a form of reorganization which leaves no indication of its former position. Even the smallest of those recrystallized units which can be clearly resolved at a magnification of 1,000 seem to bear no directional relationship to the preëxisting banded structure. This, in all probability, indi-

[11] W. Rosenhain: Der Kristallisierte und der amorphe Zustand der Metalle, *Internationale Zeitschrift für Metallographie*, vol. v, p. 65–106 (1913). A bibliography of the earlier papers by Beilby is given on p. 103 of Rosenhain's paper.

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cates that the earliest visible stages of reorganization represent conditions which exist after considerable coalescence and rearrangement have occurred, so that the first movements away from the banded structure are obscure.

It is clear that while the etch bands probably are indicative of severe dislocation, or development of inner surfaces of marked discontinuity in the general direction of one favored set of cleavage or slip planes, the absence of inner surfaces corresponding to displacements in a crosswise direction is not necessarily indicated by the lack of markings in this direction. In this connection, we should recall the entire lack of markings in metal which has received very little reduction and yet is susceptible to recrystallization. From the facts available, it is fair to conclude that the etch bands in severely deformed metal represent the development of innumerable inner surfaces of discontinuity which are highly specialized and continuous in some prevailing direction, so as to give rise to selective etching effects.

The central area of Fig. A, Plate I, shows a number of recrystallized units which seem to have grown out of the surrounding matrix of indefinable structure. This matrix no longer contains etch bands and it seems to represent a condition intermediate between visible recrystallized units and visible strain effects. In Fig. C of the same plate, we can see where the etch bands of a large, light grain in the upper left-hand corner have broken down along a jagged boundary, to the right of which the altered and indefinable matrix occurs, along with a quantity of reorientated units. It may be remarked that the first portions of metal to recrystallize are almost always in regions of intersection between a number of grains, as is the case in Fig. C. Since recrystallization occurs at successively lower temperatures with increasing reductions, it appears that these regions are regions of maximum strain, which would, indeed, be indicated by the conflicting systems of etch bands and the oppositional nature of the forces in these regions. Figs. B and D, Plate I, taken from a piece of metal which had just begun to soften, show the earliest effects which can be distinguished under a magnification of 1,000. In Fig. B, a number of minute and not very clearly distinguishable recrystallized units may be seen in regions where the etch bands have degenerated into a matrix whose structure is not dissolved. In Fig. D, the etch bands have partially broken down, while no reorientated units are distinctly shown.

We have thus far dealt with the early stages of recrystallization in metal which has received a comparatively heavy reduction by rolling. After such reduction, recrystallization begins, as we have seen, in the more severely strained areas at low temperatures, the exact values of which in any case depend upon the degree of reduction and the time of exposure. As the temperature of anneal increases under otherwise uniform conditions, additional etch bands break down, or granulate; the

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existing recrystallized units coalesce and grow; and, at no very elevated temperature, the entire preëxisting grain structure becomes replaced by a secondary, refined structure of rather uniform character. The condition which results after such a series of changes is illustrated by Fig. D, Plate X, corresponding to a reduction of 25 per cent.; an annealing temperature of 550°; and a period of 30 min.

Early Stages of Recrystallization in Material which has Received a Light Reduction

A few special experiments were made for the purpose of revealing the characteristics of recrystallization in metal which had received very light reductions. In these experiments, the reductions were not measured, but they were brought about by pressure at right angles to a polished surface; so some idea of their intensity and effects could be gathered from the changing appearance of the polished surface with its slip bands and other indications of movements in the surface region. Furthermore, in every observation, the specimen was placed in the same position on the stage of the microscope, so that the appearance of the same portion of the surface could be compared after various operations. In order to enhance the value of such comparisons, a very coarse grain was developed at the start by annealing at a high temperature. By this precaution the necessary operation of repolishing did not result in total obliteration of grains which were previously visible at the surface. It is obvious that such procedure will not enable us to refer all changes to grains previously seen on the surface, since any removal of the surface will reveal new grains at certain places, *i.e.*, where the subjacent grain boundaries were not far below the first surface. Some advantages are, nevertheless, to be derived from this method of comparison.

Space can be given to only a few of the large number of photomicrographs obtained. These illustrations, together with a brief description of the results in general, will serve to characterize the recrystallization phenomena.*

The surface-effect of a moderate amount of strain after anneal at 750° is shown in Fig. A, Plate III. Slip bands are visible in all of the grains. The characteristic angular relations between slip bands where twinning planes occur, which has been illustrated in many published photomicrographs, can be seen in a number of places. The strain has been sufficiently pronounced to produce cross-slips in a number of the grains.† There has been relatively little movement of the grains with respect to one another. As previously mentioned, we are unable to obtain positive

* Various brass mixtures were used in these experiments; the photomicrographs shown were obtained from a sample of admiralty mixture containing 72.53 per cent. Cu; 0.07 per cent. Fe; 0.04 per cent. Pb; and 1.06 per cent. Sn.

† Unfortunately, much of the detail of this micrograph has been lost in reproduction.

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evidence of recrystallization after annealing this specimen. Fig. B, Plate III, shows the repolished surface after anneal at 615°. Most of the grains visible in Fig. A can be identified in Fig. B. Some additional twinning bands can be seen, but this may be due to the difference of several thousandths of an inch between the two surface planes. If any crystallization has actually occurred it has only resulted in a slight increase in the number of grains in the affected region. It may be noted that the temperature of anneal, although rather high (615°), is more than 100° below the temperature of anneal before straining (750°). It is, of course, possible that a temperature of anneal in the neighborhood of this latter value would bring about distinguishable recrystallization. On the other hand, the preëxisting structure, irrespective of any alteration by strain in this experiment, might undergo some change at a temperature in the vicinity of the original anneal and we omitted the experiment on account of the uncertainties attached to it. We believe that, in the case of stresses which merely suffice to develop the ordinary slip bands but do not produce inner surfaces of another character, no recrystallization will occur on any annealing treatment purely as a result of these movements of translation. It is our impression that the numerous inner surfaces of slip, which are indicated by the close order of slip bands at the outer surface, are healed on annealing without reorientation in any grain.

The same specimen is shown in Fig. A, Plate IV, after strain of more severe character. In this case, the grains have moved considerably with respect to one another, and slip within one and the same grain has also contributed not a little to the existing surface irregularities owing to more extended movement in some parts of the grain than in others. These surface irregularities account for the blurred appearance of the surface in the micrograph; it is not possible to bring all parts into focus at once. On account of this difficulty, comparison between this and the annealed structure shown in Fig. B is more satisfactory at the microscope, where adjustments may be changed to suit any part of the surface, than through the medium of photomicrographs. It is apparent, however, particularly in the lower portion of the surface represented, that the number of grains has increased on annealing; the inner surfaces have been obliterated by reorganization into smaller grains. In the upper part of the figures, where strain was least severe, at least one grain common to both may be seen.

Another specimen, strained somewhat more severely than the last, is represented in Fig. C, Plate IV. The companion micrograph, Fig. D, shows the corresponding surface after anneal at 550°, a temperature about 65° lower than that adopted in the previous case. Here a large, banded grain near the center of the micrograph seems to correspond with one in about the same position in Fig. C, while groups of smaller grains in the neighborhood correspond to reorganization of the much larger primary grains shown in Fig. C.

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Experiments of this character were continued both in the direction of increasing the strain and decreasing the temperature of anneal. With regard to the former, it may be asserted that more severe strain decreases the size and increases the number of separate units which may be seen after a given annealing treatment. In other words, more uniform structures result when the metal is more severely strained. With regard to the latter, if the specimen is strained to about the same extent as that shown in Figs. A and C, Plate IV, it will not give evidence of recrystallization at temperatures much below 500°. The last micrograph of annealed material, viz., Fig. D, Plate IV, therefore represents very early stages of recrystallization in material which has received a very light reduction, and may be compared with Figs. A and C, Plate I, which represent the very early stages of recrystallization in material which has received a heavy reduction.

From the standpoint of potential recrystallization, the chief distinction between metal which has received a heavy reduction aud metal which has received a light reduction lies in the vast difference between the number of inner surfaces which have developed in the two cases. These inner surfaces cannot remain permanently in existence under changing conditions and, as the temperature is raised, they give way to more stable configurations in accordance with equilibrium requirements which take into account their number, character, and distribution, along with the other variables encountered. Two neighboring particles with separate surface boundaries will tend to coalesce, or flow together, under the influence of forces which operate to reduce the surface area and surface energy (*i.e.*, to reduce the number of separate surfaces) to a minimum. There are oppositional forces which bring about a certain degree of firmness, or rigidity, of the grain-substance, or determine directional properties which each particle tends to retain and which resist coalescence, but all of these forces vary in intensity with the temperature and, at any given temperature, there will be a condition of equilibrium beyond which further decrease of surface area is effectively opposed by the forces in question.

This is a form of equilibrium which renders the size of the particles, or the number of inner surfaces, chiefly a function of the temperature. In order that reorganization within the grain substance may proceed at a low temperature, there must be a high order of development of inner surfaces, as in the case of metal which has received a heavy reduction. Small recrystallized units may then develop at the expense of preëxisting inner surfaces. Where the metal has received a light reduction, however, there can be no development of very small recrystallized units, since there are no inner surfaces of this order; the first units to recrystallize will be comparatively large, as was noticed in the case of Fig. B, Plate IV, and they will not be able to recrystallize until a comparatively high temperature is reached.

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Much valuable material bearing upon the phenomena of recrystallization after strain, and the principles involved, has been contributed by Heyn,[12] with illustrations drawn chiefly from copper and iron, and by Tammann,[8] with particular relation to his translation theory of plastic deformation One of us[13] has lately attempted to give an outline of the relations between the annealing variables, the degree of reduction of cold-working, and the structural characteristics in connection with the study of some ancient bronzes exhibiting a wide variety of structures. Some of this material will be used at this point for the purpose of introducing a discussion of Plates V to X which illustrate the structures obtained in the case of cartridge brass by systematic variation of the annealing temperature and the degree of reduction.

Principal Relations Between Deformational Treatment and Structural Characteristics after Anneal

The accompanying diagram (Fig. 2) will serve as a basis for the ensuing remarks. This diagram is based upon the assumption that the mean size of the new grains which form at any given temperature by coalescence of existing grains or by local breakdown of internal surfaces within the original strain-hardened grains is determined by the temperature of anneal. Thus, the size of the recrystallized grain will be some function of the annealing temperature, and the conglomerate will be composed of recrystallized grains along with unrecrystallized fragments of the mechanically altered grains. Such structures are commonly observed in strained metal which has not been annealed sufficiently to cause complete recrystallization throughout the mass, *e.g.*, Fig. B, Plate XI. The entire field is traversed by a set of lines, *ab*, *cd*, etc., which cut any vertical line into a number of segments intended to represent the mean number of recrystallized grains which would normally be counted along a unit length upon the prepared specimen after anneal at the corresponding temperature. A linear relationship is assumed in this construction, but obviously any experimentally proven relationship between grain size and annealing temperature may be depicted in similar manner.

By cold-working metal of definite grain characteristics, *e.g.*, uniformly recrystallized metal of the mean grain size which would be produced by complete anneal at T_9, the grain structure is broken down, inner surfaces develop and new, latent grains of a lower order of size and stability are formed. As already pointed out, these secondary particles cannot be distinguished under the microscope. Etching peculiarities, diminishing

[12] E. Heyn: Martens-Heyn, *Materielenkunde für den Machinenbau*, Part II-A, Chapter IV (Berlin, 1912).

[8] *Zeitschrift für Elektrochemie*, vol. xviii, p. 584 (1912); also *Lehrbuch der Metallographie* (Leipzig, 1914), Chapter 1, C.

[13] C. H. Mathewson: A Metallographic Description of Some Ancient Peruvian Bronzes, *American Journal of Science*, vol. xl, pp. 525–616 (1915).

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contrast, lines of deformation, etc., are indeed visible when deformation has been severe, but no direct indication of the size or shape of the ultimate secondary units can be obtained. As far as direct observation goes, these new structure elements are latent only. Upon annealing, they assert a modified individuality wherein mutual readjustment and coalescence cause them to become visible units with definite orientation and the dimensions characteristic of the annealing temperature adopted.

In order that the annealing effect may be felt at a given temperature, it is clear that the mechanical destruction of the original grain must have been sufficiently pronounced to produce fragments inferior in size to the

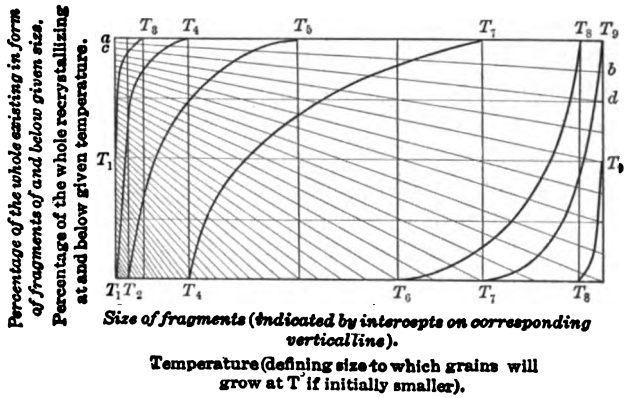

FIG. 2.—DIAGRAM IN WHICH BOTH A CONJECTURED FRAGMENTATION OR FRAGMENTAL RESOLUTION OF THE ORIGINAL GRAIN * BY DEFORMATIONAL TREATMENT AND THE PERCENTAGE OF STRAIN-HARDENED MATERIAL WHICH MAY BE EXPECTED TO RECRYSTALLIZE AT ANY GIVEN TEMPERATURE ARE REPRESENTED ON A SINGLE CURVE FOR A GIVEN DEGREE OF DEFORMATION.

recrystallized grain characteristic of this temperature. Thus, if the T_9 grain previously cited is to recrystallize at T_5, it must have been artificially reduced by deformation to a point where the grain size characteristic of T_5 can develop. In general, the grain fragments produced by deformational processes will vary widely in size, so that certain of them will be able to coalesce below T_5, while others will remain unaffected at this temperature. The particular factors which apply in any given deformational process will combine to determine a curve of "*fragmental resolution of grain*" * for this particular process in which the cumulative percentage of fragments below a given size appears as a function of the size of the fragments.

* We have introduced this phraseology to aid in defining the disintegrating effect of cold-working without regard to the specific nature of the fragments, or particles, formed. Most theories freely admit that subgranular units possessing some degree of individuality are produced when the metal is stressed to the point of permanent deformation. Fragmental resolution of grain here signifies the distribution, according to quantity and size, of (invisible) fragments produced by a given form of deformational treatment.

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The curve T_4T_7 is a hypothetical curve of this sort and signifies, in the present case, that, by deformation of the original T_9 grain approximately 65 per cent. of the material has been sufficiently reduced (by internal slip) to recrystallize by the time the temperature T_5 is reached. At T_5, then, 65 per cent. of the material will have reached the mean grain size characteristic of this temperature, while 35 per cent. will remain unannealed.

Little can be said relative to the distribution of the unannealed fragments and the recrystallized grains. In general, the former occurs in patches of irregular outline and, when present in comparatively small amount, as in the present case, cannot usually be distinguished from the latter. When present in large amount, however, the patches are large, compared with the size of the recrystallized grains, and often bear evidence of strain. Such a condition is shown in Figs. A and C, Plate I. Here large patches of metal bearing distinct lines of deformation can be seen along with small recrystallized grains.

The first formation of recrystallized grains usually occurs at the boundaries of the parent grains, since, at such localities, maximum unhomogeneity of the lines of force may be expected and, consequently, maximum fragmental resolution of grain.

As the degree of deformation increases, whether by rolling, drawing, hammering, etc., the curve of fragmental resolution recedes in the direction of maximum resolution, a specific condition for each metal or alloy beyond which further destruction of grain will bring about fracture. Recession in this direction is indicated by the set of curves, T_2T_5, T_1T_4, and T_1T_3. As maximum resolution is approached, the abscissa range covered by the curve becomes narrower, since large fragments give place to smaller ones and the size of the latter cannot be reduced without limit. Ultimately we would reach a condition of uniform resolution into fragments of minimum size. It is improbable that such a condition can be realized without overworking the metal to the point of manifold fracture. Thus the limiting curve, T_1T_3, is drawn to represent a 50 per cent. resolution into fragments of minimum size.

There is, for each metal or alloy, a minimum temperature at which recrystallization will start from a condition of maximum resolution into fragments of minimum size. This temperature is represented by T_1 in the present diagram. One-half of the metal deformed according to the curve T_1T_3 would recrystallize at T_1. At the higher temperature, T_3, all of the metal deformed according to the curve T_1T_3, about 85 per cent. of the metal deformed according to T_1T_4, about 35 per cent. of the metal deformed according to T_2T_5, but no part of the metal deformed according to T_4T_7, would recrystallize. At T_7, the grain of all four would be identical and would grow uniformly beyond this temperature. These curves all represent severe deformation, in that the percentage of coarse frag-

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ments is low compared with the percentage of fine fragments. In the alpha brasses, such a condition occurs when the reduction by rolling or hammering is carried beyond about 15 or 20 per cent. Above 550°, provided the annealing period is not confined to a few minutes, alpha brass of definite composition will give rather uniform grain characteristics whatever the extent of previous reduction beyond the minimum value given. Hence, we may draw certain conclusions relative to heat treatment from these grain characteristics without intimate knowledge of the previous mechanical treatment.

Turning now to the effect of anneal upon metal which has received very light deformational treatment, it appears that the curves of fragmental resolution will here assume a somewhat different form, in that the percentage of coarse fragments will very likely exceed the percentage of fine fragments. This condition is represented by the curves, T_6T_8, T_7T_9, and T_8T_9. Moreover, none of the fragments are likely to be very small and, accordingly, recrystallization will not start until moderately high temperatures have been attained. Recrystallization according to the curve T_6T_8 starts at T_6, but is confined to small portions of the mass until the temperature rises toward T_8. At the latter temperature, the entire alloy is recrystallized. According to the curve, T_8T_9, the extremely light deformation indicated has left 50 per cent. of the original T_9 grain unaltered, while the remainder has been broken down sufficiently to cause initial recrystallization at T_8, a temperature not greatly below the original annealing temperature. At T_9, the metal has assumed its original grain characteristics and growth will continue normally as the temperature is further elevated. The curve, T_7T_9, represents an intermediate condition which may easily be interpreted.

In cases of light deformation, recrystallization cannot always be detected under the microscope, since the recrystallized grains are not greatly inferior in size to the original grains. In this connection, it should be noted that a section through the conglomerate shows large and small grain sections whatever the true size of grain, since a grain may be cut at any point according to its position with regard to the cutting plane. This renders it difficult to detect recrystallization except where the new grains are considerably smaller than the original grains and are sufficiently numerous to form groups of characteristic appearance.

The curves of fragmental resolution are purely hypothetical and represent in a general way what seem to be reasonable characteristics of the ordinary deformational processes. The precise form of any curve will be determined by the nature and intensity of the deformation sustained by the metal.

It is clear from the discussion thus far that the form of a curve of fragmental resolution determines the form of a curve representing the cumulative percentage of metal recrystallized as a function of the tem-

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perature. Without seeking to establish the exact relationship between these curves, it may be assumed that they are substantially identical in form, the former representing an obscure condition and the latter a visible effect. By careful counting under suitably chosen magnification, the approximate form of some of these curves may be determined. The present discussion is merely intended to convey a rational conception of deformational and recrystallization phenomena from a purely qualitative standpoint. As far as the present brasses go, it is possible to detect certain characteristic effects, such as an early or a late stage of recrystallization after severe deformation, without inquiring into the details of the deformational treatment (degree of deformation, etc.).

It may be remarked that a diagram of this sort may be drawn to represent the condition developed by anneal during a purely arbitrary period of time, or by anneal during such a period as may be necessary to bring about an equilibrium effect. The latter (ideal) condition was kept in mind throughout the foregoing discussion. The former condition would entail displacement of all recrystallization curves without affecting the general principles involved.

Owing to the difficulty of distinguishing between recrystallized and unrecrystallized units, a count of the grains in partially recrystallized conglomerates is not especially useful or significant. It is possible to obtain at first an increase and, later on, a decrease in the number of visible grains as the period of anneal at constant temperature, or as the temperature for constant period of anneal, increases. This does not signify that the individual grains first disintegrate (contrary to thermodynamical requirements) and later coalesce, but that the original strain-hardened grain fragments (each of which is counted as a single unit) are rapidly developing secondary grains in the early stages of the anneal; while later, coalescence alone constitutes the predominant factor.

Structural Changes Developed in 70/30 *Cartridge Brass by Uniform Anneal at Six Different Temperatures within the Range,* 350 *to* 800°, *after Reductions of* 2, 4, 8, 12, 15, *and* 25 *Per Cent., Respectively*

The material used in these tests came to us in the form of cold-rolled strips 8 in. long by 0.5 in. wide by 0.1 in. thick. They had received a reduction of six numbers, according to the B. & S. gage (50 per cent. reduction of sectional area), after the last anneal, and gave the following tests when in this condition:

Tensile strength	87,750 lb. per square inch.
Reduction of area	38 per cent.
Elongation, 1 in.	13 per cent.
Elongation, 3 in.	5.3 per cent.
Hardness by scleroscope,	36.5 per cent.

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Our analysis of the material showed 70.16 per cent. of copper, 0.06 per cent. of iron, and a trace of lead.

Before rolling to the adopted values of reduction, these strips were brought to a condition of nearly maximum softness (7 to 7.5 by scleroscope) by annealing for a period of 20 min. at 800°C.

A small hand rolling mill was used in obtaining the series of six reductions. Very careful measurements of gage gave the following values: 2 to 2.3 per cent., 4 per cent., 7.7 per cent., 11.8 per cent., 15 per cent., and 24.3 to 26.6 per cent. Variations in the first case were due to the difficulty of uniformly regulating the very slight reduction desired (the thickness was reduced only 0.002 in.), while variations in the last case were due to an exceptional variation in the thickness of the original strip.

The rolled strips were then cut up into pieces about 1 in. long which were thereupon annealed at each of the temperatures, 350°, 450°, 550°, 650°, 750°, and 800°, in groups containing one piece to represent each reduction. The anneals were at constant temperature (± 5°) for a period of 30 min. and the preheating period varied between 8 and 11 min. in the different cases. The following annealing record at 550° will give a fair idea of this part of the work:

Preheating Period	Time	Annealing Period	Temperature, Degrees Centigrade
	3.52		525, in center of furnace.
	3.53		Specimens entered.
11 min.....	3.59		517, in center of group.
	4.04		525, in center of group.
	4.15		527, in center of group.
	4.20	....30 min	525, in center of group.
	4.34		524, in center of group.

The above average annealing temperature of 525°, as indicated by the thermocouple, corresponds to 550°, as corrected by interpolation on the calibration curve.

Photomicrographs taken from all samples, with the exception of those which received a 30-min. anneal at 800° following the cold-work, are assembled in Plates V to X. It is superfluous to represent the regular anneal at 800°, since the characteristics of this anneal are the same in all cases and the first photo of each plate, taken after reduction of the material which had previously received an anneal at 800°, will serve to illustrate the principal features of this high-temperature anneal.

Each plate shows the rolled structure after a given reduction, followed by the annealed structures developed in the treatment of this particular material. Not all of the micrographs will show the same quality of contrast or the same degree of surface excellence acquired in the preliminary operations of polishing and etching. Differences of this character are, of course, to be disregarded in making structural com-

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parisons. A particular effort was made in each case to select a spot of average and typical appearance when preparing the micrographs.

Turning now to Plate V, which shows the worked and annealed structures corresponding to a trifling reduction of 2 per cent., we are unable to detect any certain evidence of recrystallization at any of the annealing temperatures. A certain number of small grain-sections are seen in all of the micrographs and will always be seen on the prepared surface of a coarse-grained conglomerate, since some of the grains will be cut in attenuated regions; there are not enough of these small sections to positively indicate recrystallization at the lower temperatures of anneal. Recrystallization would be even less evident at the higher temperatures, 650° or 750°, since, as we have indicated in the preceding section, the reorientated grains which form at these temperatures are comparatively large and would ordinarily be overlooked in the midst of the 800° conglomerate. On the basis of the generalizations already introduced (preceding section), we would expect recrystallization to begin at the higher temperatures after such a trifling amount of reduction, and the fact that very little of the hardness acquired as a result of the mechanical treatment is lost on annealing until the higher temperatures are reached (see next section) confirms this conclusion.

After a 4 per cent. reduction, the microscope is more effective in revealing the early stages of recrystallization. By examination of the micrographs shown in Plate VI, it is seen that a distinct refining of the grain has resulted from the 650° anneal. The micrograph corresponding to anneal at 450° suggests the possibility of recrystallization, but subsequent examination of the entire surface of the specimen has convinced us that, owing to an unfortunate choice of position, the grain shown in this micrograph is below the average in size. From previous considerations, we argue that recrystallization at 650° cannot be seen in Fig. E, Plate V, corresponding to a 2 per cent. reduction, because of the comparatively small number of inner surfaces which are of the proper character and dimensions to coalesce at 650°, while in Fig. E, Plate VI, corresponding to the 4 per cent. reduction and the same annealing temperature, there are a sufficient number of such inner surfaces to yield groups of small grains, after readjustment, which, along with the residual fragments of the parent grains, form a conglomerate that can readily be distinguished from the original aggregate of much coarser appearance.

It is clear that while this is a conglomerate—in the sense that it is composed of recrystallized units of normal average size for this temperature of recrystallization together with residual fragments of the original grains—an anneal under similar conditions, in the case of metal which had received a sufficiently heavy reduction to produce fragmental resolution into inner surfaces entirely disappearing at 650°, would yield, not a conglomerate in this sense, but an aggregate of grains possessing the

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normal average size corresponding to 650°. The latter type of structure is represented by Fig. E, Plate X, and by Fig. E, Plate XI, which correspond to anneal at 650°, following reductions of 25 and 50 per cent., respectively. From a practical standpoint, to produce uniformity of structure (and uniformity of properties) at a given temperature of anneal, the metal should have received a certain minimum degree of reduction, which, in the present case, is in the neighborhood of 20 per cent. Other deductions of this sort may be made on the basis of the accompanying photomicrographs.

For obvious reasons, the first positive indication of recrystallization in a set of micrographs showing changes of structure as we proceed from one temperature of anneal to another does not represent the earliest actual recrystallization. There is an apparent lag, owing to our inability to distinguish properly a few recrystallized units from the rest of the aggregate when these units are not very small in comparison with the parent grains, *i.e.*, when the reduction has been light. In the case of heavy reductions, this difficulty is not a serious one. We have already mentioned in an earlier section of this paper that a drop of one or two points on the scleroscope scale can be detected in the form of recrystallization by examination under high power when the reduction has been severe. With a reduction as low as 2 per cent. (Plate V) the total drop of about three points throughout the entire annealing range (350° to 800°) could not be detected in this form, as we have lately seen. It is clear that no advantage would be derived from the use of higher powers in this connection, since there are no minute units to be seen.

It is not necessary to give a detailed discussion of the other plates of the set (Plates VII to X). It will be noticed that, as the degree of reduction increases, recrystallization can be distinguished at successively lower temperatures and the first recrystallized units become smaller and smaller, as is required by the theory. Table III gives a summary in this respect.

TABLE III

Percentage Reduction	Temperature of Earliest Visible Recrystallization at 85 Diameters after ½-hr. Anneal
2	.....
4	650°
8	550°
12	550°
15	450°
25	350°
40	275° to 300° (× 1,000)

Recrystallization at 350° after a reduction of 25 per cent. is not clearly shown in the corresponding micrograph (Fig. B, Plate X). A few groups

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of very small units may be seen† when the micrograph is examined carefully. High-power examination leaves no doubt of the actual recrystallization, although only a small part of the metal has become affected. Cartridge brass of this composition, which has been reduced 50 per cent. and then annealed for 30 min. at this temperature, shows abundant recrystallization with residual patches of strain-hardened metal. Fig. B, Plate XI, is thoroughly representative of this condition, although it relates directly to a mixture of somewhat different composition.

The characteristics of rolled structures have already been illustrated (Plate II) and briefly described. Additional illustration is afforded by Fig. A of each of the Plates V to X inclusive. Increased curvature of the twinning bands is noticeable as the reduction increases, while the etch bands first attain prominence in Fig. A, Plate X, corresponding to a reduction of 25 per cent.

Distribution of the Total Drop in Scleroscopic Hardness in Each of the Above Series According to 100° *Temperature Increments of Anneal*

The small pieces of metal from which the photomicrographs described in the preceding section were made furnished suitable material for scleroscopic tests. After removing the top and bottom surface layers by light grinding with moderately fine emery, from 10 to 20 readings were taken for each piece. Variations of one, or even two points were commonly encountered in the same piece, particularly in the harder pieces. The averages, along with calculations of the comparative drop in hardness from temperature to temperature, are given in Table IV.

TABLE IV

Reduction by Rolling	2 Per Cent.	4 Per Cent.	8 Per Cent.	12 Per Cent.	15 Per Cent.	25 Per Cent.	50 Per Cent. **
Scler. No. after rolling	9.7	12.7	15.2	17.3	21.5	27.4	38.2
Scler. No. after 350° anneal	9.6	12.7	14.6	16.6	19.5	23.6	16.8
Drop between 0° and 350°, per cent.*	3.8	0. 0	7.5	7.0	14.6	19.4	68.6
Scler. No. after 450° anneal	9.5	12.0	13.2	13.6	13.0	13.7	13.5
Drop between 350° and 450°, per cent.*	3.8	13.5	17.5	30.0	47.4	50.5	10.6
Scler. No. after 550° anneal	9.3	10.9	11.1	9.8	9.1	11.8	10.5
Drop between 450° and 550°, per cent.*	7.7	21.2	26.2	38.0	28.5	9.7	9.6
Scler. No. after 650° anneal	8.7	9.3	8.6	8.3	8.6	10.2	8.0
Drop between 550° and 650°, per cent.*	23.1	30.8	31.2	15.0	3.6	8.2	8.0
Scler. No. after 750° anneal	7.7	8.0	7.3	7.5	8.3	8.2	7.0
Drop between 650° and 750°, per cent.*	38.5	25.0	16.2	8.0	2.2	10.2	3.2
Scler. No. after 800° anneal	7.1	7.5	7.2	7.3	7.8	7.8	.....
Drop between 750° and 800°, per cent.*	23.1	9.6	1.2	2.0	3.6	2.0	.....
Total drop, points	2.6	5.2	8.0	10.0	13.7	19.6	31.2

* Given as a percentage of the total drop.

** The original cold-rolled material was used for these tests. It differs from the other material in that no preliminary anneal at 800° was introduced; pieces were cut from the original strips, which had received 50 per cent. reduction in the mill, and were given the successive anneals under the uniform conditions observed in all of the work.

† This observation applies to the original micrograph; owing to loss of detail in reproduction it may not be possible to see these units.

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The proportion of the total drop in hardness which is realized during each temperature stage of the anneal is plotted in Fig. 3 for each series (by reductions), as a function of the temperature which completes the given stage.

We cannot assert that the values given in Table IV and the correspond-

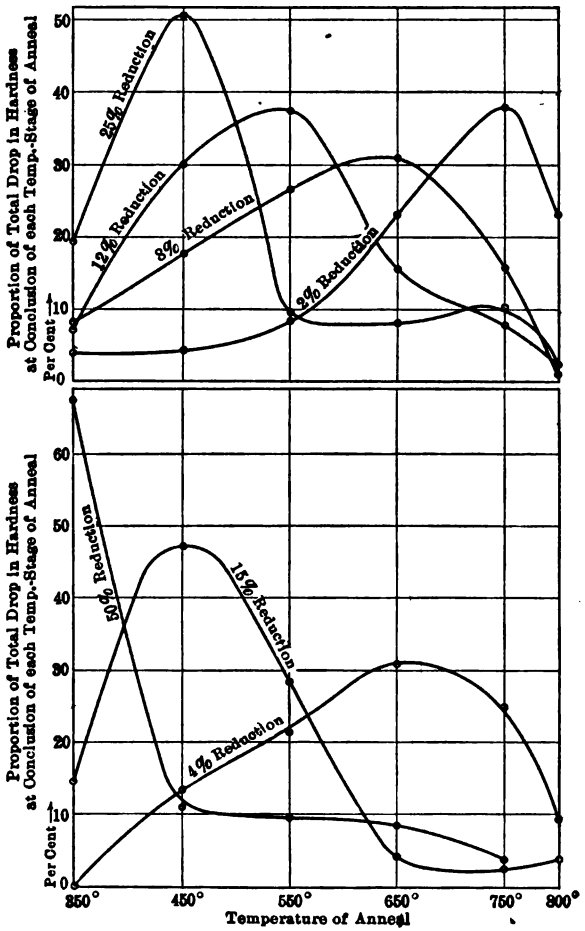

FIG. 3.—DISTRIBUTION OF HARDNESS DROP ACCORDING TO SUCCESSIVE TEMPERATURE STAGES OF ANNEAL-CARTRIDGE BRASS (70/30). THE PERCENTAGE REDUCTION IS INDICATED ON EACH CURVE.

ing curves of Fig. 3 are accurate to any high degree. Many of the drops were in the neighborhood of one point, according to our measurements. Our experience indicates that it is not reasonable to expect anything more than approximate hardness indications of this order. Thus, the entire curve corresponding to a 2 per cent. reduction is of an approximate nature and the same applies to portions of the other curves which corre-

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spond to slight changes in hardness. The principal indications of these curves are, however, of value and may be freely used to supplement the information gained from the study of the photomicrographs.

First of all, we observe that there is a maximum on each curve which unquestionably points to particularly active recrystallization in this region and bears favorable comparison with the corresponding structural evidence. For example, the maximum on the curve representing a 25 per cent. reduction occurs in the neighborhood of 450°, which is also true of the curve representing a 15 per cent. reduction, and we have previously observed that the earliest prominent development of recrystallized units occurred at this temperature in both cases (Fig. C, Plate X and Fig. C, Plate IX). The curve representing a 12 per cent. reduction possesses a maximum in the neighborhood of 550° and we first observed recrystallization at this point (Fig. D, Plate VIII). In the curves representing 8 and 4 per cent. reductions, the comparatively flat maxima are located (probably) to the left and to the right of 650°, respectively, while we were able to detect recrystallization at 550° in the former (Fig. D, Plate VII) and at 650° in the latter (Fig. E, Plate VI). The curve representing the lowest reduction of the series, 2 per cent., rises to a maximum at the highest temperature of all, viz., about 750°, but, for reasons which have been set forth, we could not detect recrystallization at this, or any other temperature.

Assuming that the drop in hardness is due chiefly to recrystallization, a restricted temperature range of rapid drop would correspond to a restricted temperature range of active recrystallization, while a wider range would correspond to more gradual recrystallization over a wider range of temperature. According to this interpretation, the curves which are based upon the higher reductions of 50, 25, and 15 per cent. and the curve which is based upon the minimum reduction of 2 per cent., all four possessing comparatively sharp maxima, represent recrystallization within a comparatively limited range of temperature, while the curves which are based upon the medium reductions of 12, 8, and 4 per cent., all three possessing comparatively flat maxima, represent recrystallization throughout the greater part of the entire range shown in the diagram. Since we can clearly distinguish only the most pronounced effects by direct observation, the above deduction is not easily verified, at least with the sole aid of the material available at the present time. Aside from all questions as to the value of the experimental results involved, this is a perfectly logical deduction from a theoretical standpoint. It signifies, in this connection: (*a*) that the lowest reduction develops a small number of widely separated inner surfaces which resolve the original grain into comparatively few fragmental units and, consequently, break down, for the most part, at a temperature not far below the temperature of equilibrium of the normal grain; (*b*) that the medium reductions develop a large number of

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inner surfaces which vary considerably in magnitude and begin to break down at a temperature far below the temperature of equilibrium of the normal grain, but continue in this process up to, or nearly up to, the above temperature; and (*c*) that the heavy reductions develop an extremely large number of inner surfaces of uniformly small size which break down at low temperature throughout a limited range. Considerations of this sort were formulated on preceding pages and idealized in Fig. 2. The curves of fragmental resolution, T_8T_9, T_4T_7, and TT_4, shown in this diagram, correspond to the respective conditions (*a*), (*b*), and (*c*), specified above.

The question naturally arises whether the relative drop in hardness from temperature to temperature indicates the relative quantity of metal which has recrystallized in the respective interval, or is possessed of some modified significance. Since the rebound of the scleroscope hammer depends not only upon the amount of penetration at the point, but upon the elastic properties of the metal as well, it is necessary, in order that the different readings may possess a strictly comparative value, that several ill-defined properties vary uniformly as the ultimate testing results on the metal vary. Even if this condition is realized, a drop of a given number of points along different sections of the empirical scale will not necessarily correspond to the same specific change in properties, although, within a limited reading range this difficulty must largely disappear. Therefore, we must proceed with caution in using the indications of the scleroscope as a quantitative measure of any structural change within the metal, such as the destruction of inner surfaces, or of any associated property, such as elastic limit, or ultimate strength. The validity of all possible relationships of this sort must be tested by experiment. Any simple conversion factor will, as a rule, apply only throughout a limited range, while satisfactory calculations over a wider range may sometimes be possible with the aid of formulas (often complicated) based upon an analysis of the graphical relationships encountered. Compare, in this connection, papers by Shepard and Porter[14] and by Shepard and Summers[15] on hardness tests of cold-rolled steel and brass, respectively.

Aside from all difficulties in the interpretation of results which originate in the choice, or application, of the testing instrument, we must distinguish between effects which are due solely to changes in the size of grain, *i.e.*, to removal of inner surfaces between natural grains, and those which are due to actual recrystallization, or removal of inner surfaces of strain.

It is not a simple matter to test the effect of grain size upon hardness without introducing the element of strain, since great variations of grain

[14] Shepard and Porter: Hardness Tests of Cold-Rolled Steel, *American Machinist*, vol. xlii, pp. 277–278 (1915).

[15] Shepard and Summers: Hardness Tests of Cold-Rolled Brass (to appear in the *American Machinist*).

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size cannot ordinarily be produced without first straining the metal and then annealing it. Such variations as may be produced by varying the rate of cooling in casting the metal (with subsequent annealing to insure homogeneity of composition), do not greatly influence the hardness. Those who have had occasion to test a variety of cast alpha brasses are aware that variations in hardness by the scleroscope never amount to more than a few points in metal of the same composition. George V. Caesar, in the Hammond Laboratory, has carefully tested the scleroscopic hardness of samples of pure copper and of the same grade of copper containing small quantities of cuprous oxide. In the annealed condition after rolling, the samples of pure copper show materially coarser grain than the samples of pure copper containing oxide, but the difference in hardness is not greater than would ordinarily correspond to the addition of the oxide. For example, the pure metal as cast gave a hardness number of 4.2, while metal containing 0.05 per cent. of oxygen, as cast, gave a number of 6. The difference of 1.8 points is due, principally, to the presence of oxide, as the grain, in the two cases, is at least roughly comparable in size. After a reduction of 66⅔ per cent. followed by anneal at 925°, the pure metal gave a hardness number of 5, while the metal containing oxide gave a number of 6.4.

Figs. D and F, Plate X, of the brass series may be used to illustrate the considerable difference in grain size in the last two cases. The hardness difference of 1.4 points appears to bear little relation to the difference in grain size, since it is comparable with the difference observed in the case of the cast metal (1.8 points). In making use of these comparisons, we assume that, owing to the uniform temperature of anneal, actual recrystallization has occurred to the same extent in both kinds of material, but that in one case the grain has grown without hindrance to normal size, while in the other case the oxide has interfered mechanically with coalescence and thus reduced the size of grain. On this basis, we conclude that changes in grain size alone are not accompanied by considerable changes in hardness.

Turning again to the curves of Fig. 3, it may be safely asserted that each curve represents a summation of two curves; one corresponding to relatively slight drops which are due to growth of grain without regard to recrystallization from the strained condition, and the other corresponding to the more pronounced drops which are due to removal of the inner surfaces of strain. When the latter effect is principally confined to a limited range of temperature, some differentiation between the two effects is possible, as in the case of the four curves representing 50, 25, 15, and 12 per cent. reductions, respectively. In these curves, the pronounced effects which occur in the vicinity of the maxima and which are combination effects, as suggested above, are superseded by comparatively slight effects in the higher temperature regions which are due mainly to growth of grain. It is clear that we cannot sharply define the point at

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which summation of the two effects first occurs in the case of any curve. Where the maximum is sharp, as in the case of the 50, 25 and 15 per cent. reductions, some approximation of this sort may be made and we have already used these indications to aid in interpreting the general characteristics of recrystallization (p. 636). The use of such curves in a quantitative sense is an entirely different matter, and would involve too many assumptions to yield results of much value. For example, in the case of the curve representing a 25 per cent. reduction, we might interpret the change in direction at 550° to indicate the conclusion of recrystallization (from a condition of strain) at this point. We cannot be sure that there is no further recrystallization, *i.e.*, between 550 and 600°, since the last stages in the disappearance of inner surfaces may be too gradual to show an end point of this character on the curve. Assuming that recrystallization ceases at 550°, however, we might use the drops in hardness from temperature to temperature up to this point to indicate the actual proportions of the metal which recrystallize in the respective interval. This involves assumptions of proportionality as affecting the general indications of the instrument (see p. 637), as well as similar assumptions covering the relative changes in hardness due to growth of grain throughout the intervals in question. It also involves a fundamental assumption that a unit drop in hardness corresponds to disappearance of inner surfaces (recrystallization) in a unit quantity of material, which can only be approximately correct since the inner surfaces are removed progressively from one stage of the anneal to another in the order of increasing size, or decreasing concentration, per unit volume. If, however, we plot the drops in hardness from temperature to temperature as cumulative percentages of the total drop up to a limiting point in the above sense, a curve is obtained which, under the above assumptions, gives some idea of the amount of material which has recrystallized at each successive temperature of anneal. In the case of the 25 per cent. reduction, the amount at 350° is 24 per cent.; at 450° it is 87 per cent.; and at 550° it is 100 per cent. The curve, T_2T_5, of Fig. 2 very nearly corresponds to these figures if we assign temperature values of 335°, 360°, 415°, and 550° to T_2, T_3, T_4, and T_5, respectively. Other curves derived in the same way assume positions which are consistent with the general arrangement shown in Fig. 2. Owing to the approximate nature of the numerical relationships shown, very little of practical value would be gained by introducing a diagram based entirely upon the present experiments. We will merely say, in this connection, that these experiments give material support to the general theory of recrystallization discussed on pp. 626 to 630.

Owing to the fact that the scleroscope "measures a very complex function of different properties, into which one particular kind of hardness

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merely enters as one factor"[16] and therefore gives readings which, when put to any particular use, must be held in close association with the primary conditions of experiment (chemical composition, etc.), this instrument has been freely criticised and often rejected when it would have given satisfaction if used with proper discrimination. Our experience with brass samples representing every condition of strain-hardening and recrystallization has led to the conclusion that the scleroscopic test is quite as effective as any other ordinary test in detecting slight changes in temper (as ordinarily understood in brass mills), while it is infinitely superior to any other test in point of convenience and rapidity. As compared with the tensile test, a difference of one point on the scale is roughly equivalent to a difference of 1,000 lb. per square inch (50 lb. actual load on a section 0.1 by 0.5 in.) of ultimate strength in strain-hardened alpha brass. It is obvious that a number of tensile tests on carefully prepared test pieces are necessary in order to detect differences of this order, while the uncertain skin effect and possible imperfections in the test piece combine to render the results open to suspicion. Slight differences in temper from point to point are clearly revealed by the scleroscope and there is no difficulty in obtaining an average which fairly represents the condition of the metal. With regard to the micrographic method, it is apparent, from previous explanations, that slight relief of strain due to recrystallization can be detected only when very small recrystallized units are formed. The scleroscope indicates such effects whatever the previous condition of strain or the nature of the prevailing recrystallization. The Brinell ball test and the Ludwig cone test are no more sensitive that the scleroscopic test and their only advantage lies in the fact that the respective results are more easily recalculated in terms of other properties.

It is interesting to note that Rose[2] has defended the scleroscopic method in the face of considerable criticism. We can but consider it fortunate that his instructive studies on annealing were based upon this convenient but highly sensitive method.

Comparison of Ordinary Physical Properties and Grain Size in Samples of Alpha Brass after a 30-Min. Period of Anneal at Different Temperatures

We have made careful measurements of the number of grains per unit area of section as a function of the temperature of anneal (constant period) in various brass mixtures. The heavy-faced curve shown in Fig. 4 is representative of these measurements and applies to a mixture containing

[16] W. Rosenhain: *Study of Physical Metallurgy*, Chapter X, p. 223 (London, 1914).

[2] *Journal of the Institute of Metals*, vol. viii, pp. 86–125 (1912).

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66.65 per cent. of copper, 0.30 per cent. of lead, and 0.08 per cent. of iron. For comparison, curves showing the hardness (by scleroscope), tensile strength, elongation, and reduction of area are given in the same figure. All of the corresponding data may be found in Table V. Representative photomicrographs are assembled in Plate XI.

TABLE V (*a*)

Nature of Anneal			Structural Detail							
Temperature, °C.	Time in Furnace, Min.	Time at Annealing Temperature, Min.	Magnification shown	Magnification of Count	No. Grains per Linear 2¾ In.		Average No. Grains per Linear Inch at 85 Diameters		$\frac{N}{N'}$	No. Grains per Square Inch at 85 Diameters
					Direction of Rolling	Direction Perpendicular to Rolling	Direction of Rolling (N)	Reverse Direction (N')		
350	45	30	85	.....	.....	.....	.....	.....	....	.....
450	45	30	85	119	35, 34, 33, 35, 33	41, 40, 37, 42, 36	17.40	19.90	0.87	346.2
550	45	30	85	85	26, 26, 27, 25, 26	29, 28, 28, 27, 26	9.40	10.20	0.92	95.9
650	45	30	85	17	39, 38, 36, 38, 37	39, 40, 41, 38, 39	2.76	2.84	0.98	7.8
750	41	30	85	17	27, 28, 28, 29, 27	28, 29, 26, 29, 30	2.04	2.06	0.99	4.2

TABLE V (*b*)

Nature of Anneal			Physical Tests							
Temperature, °C.	Time in Furnace, Min.	Time at Annealing Temperature, Min.	Initial Area	Final Area	Per Cent. Red. Area	Elongation in 1 in., Per Cent.	Elongation in 3 in., Per Cent.	Breaking Strain, Pounds	Tensile Strength, Lb. per Sq. In.	Scleroscope Number
350	45	30	0.4905×0.102 =0.0500	0.350×0.065 =0.0228	54.4	52	38	2,890	57,800	19.5
450	45	30	0.511×0.100 =0.0511	0.340×0.060 =0.0204	60.0	61	48	2,730	53,425	16.3
550	45	30	0.508×0.101 =0.0513	0.330×0.060 =0.0198	61.4	71	58	2,515	49,025	12.8
650	45	30	0.5145×0.1005 =0.0517	0.330×0.065 =0.0215	58.4	78	68	2,320	44,874	9.5
750	41	30	0.494×0.102 =0.0504	0.0335×0.065 =0.0218	56.7	77	67	2,130	42,261	7.5
Unannealed			0.4935×0.102 =0.0503	0.0410×0.080 =0.0328	34.8	15	6	4,070	80,919	34.5

The original material was given a final reduction of 50 per cent. by rolling in the mill. Its grain characteristics (Fig. A, Plate XI) indicate that it had been annealed in the neighborhood of 650° previous to this reduction. Physical tests on the strain-hardened metal are given in Table V (*b*).

Laboratory anneals were conducted as described on pp. 611–612. In

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Table V (*a*), the time during which the temperature, as measured in the drilled test pieces, remained sensibly constant at the annealing temperature is given for each set of two bars; also the total time in the furnace.

The number of grains per unit area was calculated from counting data. In counting, three arbitrary standards of magnification were adopted, viz., 17, 85, and 119 diameters (ratio, 1:5:7). For general purposes, each annealed specimen was photographed at 85 diameters. Not all such photomicrographs are suitable for use in counting, owing to the relatively small number of grains shown in the highest anneals

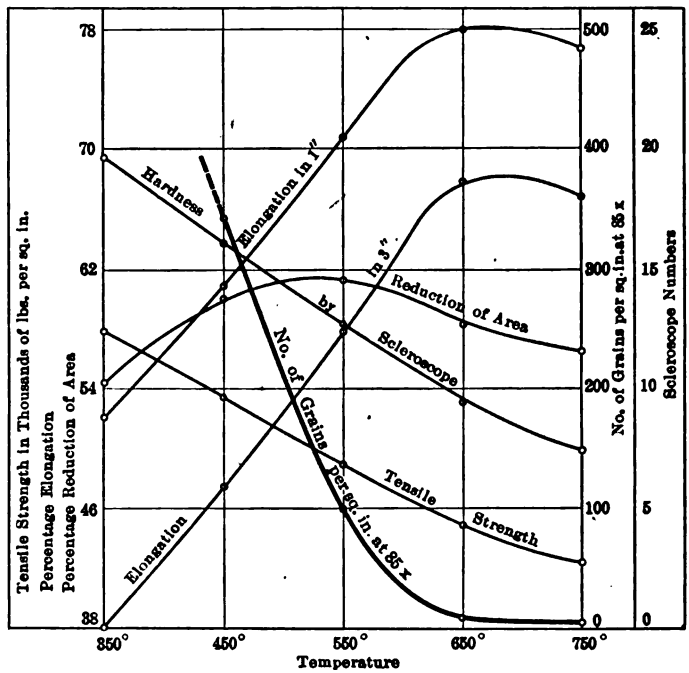

FIG. 4.—PHYSICAL PROPERTIES AND GRAIN SIZE AFTER 30-MIN. PERIOD OF ANNEALING. PREVIOUS REDUCTION, 50 PER CENT. BRASS MIXTURE CONTAINING 66.65 PER CENT. COPPER; 0.30 PER CENT. LEAD; AND 0.08 PER CENT. IRON.

and the extremely large number shown in the lowest anneals. It was deemed advisable to include at least 25 grains in each linear count and, by proper choice of one of the three above magnifications, this could be done throughout the entire series of high and low brasses. No attempt was made to count the grains developed as a result of any anneal below 450°C. Plate XI shows that below this temperature the annealed structure is made up of very minute recrystallized units along with coarse unrecrystallized fragments of the original strain-hardened grains. Aside from the mechanical difficulty of counting in such cases, we would hardly be able to make consistent use of the resulting data. The microscope

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may be used most advantageously in the case of extremely low-temperature anneals to indicate approximately how much of the strain-hardened metal has been altered by recrystallization.

We may say, in general, that the effective counting range of 450° to 750° (or higher) lies within the "zone de revenus," according to Grard,[1] or the temperature region in which a slow recovery of the extreme properties conferred by anneal is effected. Recrystallization has already spread throughout the grain substance at lower temperatures and the present changes in properties are due to growth of grain alone, or to this in combination with disappearance of the grosser inner surfaces of strain (see discussion in the previous section). This range covers the ordinary annealing treatment of brass in a practical sense.

The method of counting was as follows: Five parallel equidistant lines were ruled in one direction and five in a direction at right angles to this upon a clear gelatine-coated glass plate. This plate was then placed upon the photomicrograph and counts were made along the lines for a uniform distance of 2¾ in. From the five counts in each direction, the average number of grains per inch in the direction of rolling and perpendicular to the direction of rolling was calculated. In many cases, the occurrence of twin-crystals adds to the difficulty of counting. In general, where twins are observed, the whole polygonal section of the twinned grain, embracing two or more bands, is more comparable in size to the majority of single grains than the separate sections of the twin would be. Hence, twinned grains are counted as a single grain. This is altogether the logical method, as the material is continuous in a twinned grain, *i.e.*, there is no possibility of any (intercrystalline) impurity occurring along a twinning plane. Undoubtedly, in many cases, where the magnification is not well chosen, or the etching is somewhat below standard, or where the whole twin appears half-tone and thus shows its variations only imperfectly, it will be difficult to distinguish between twins and single individuals. On the ground of expediency, since about 300 grains are counted upon each photomicrograph, it is not advisable to spend a large amount of time in studying each doubtful case. For the same reason, it is not feasible to inquire too closely into the manner of arrangement of the grains; whether, owing to an irregular interlocking arrangement, the same grain is counted twice upon the same line, etc. In the case of low-temperature anneals, the larger grains may often represent fragments of the original grain-substance, containing within their boundaries smaller, recrystallized grains. The present method does not take proper account of such conditions. There is, of course, no conceivable method of counting the unaltered and the recrystallized areas separately, since one cannot always be distinguished from the other.

[1] *Revue d'Artillerie*, Feb.–Apr., 1909.

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Relative to the general significance of the count, we can conceive of no ready method which will give the actual size of the grains, since they are cut by any plane into large and small sections as determined by their shape and orientation. As Gulliver remarks,[17] when an etched area is measured and divided by the number of grains counted, the result gives only the "average area of a number of co-planar sections." It is, however, true that such procedure gives correct information as to the quadratic degree of dispersion, or the degree of discontinuity at the given location, and if a sufficiently large number of grains are counted in material of the requisite homogeneity the resulting magnitude is scarcely less definite in physical significance than a magnitude describing the actual number of grains per cubic unit of material. It is, of course, incorrect to base calculations of this sort upon measurements in one direction, since there may be a prevailing extension of grain growth in some particular direction owing to mechanical treatment or some obscure cause.

The counting data given in Table V (*a*) includes (columns 6 and 7) the actual number of grains counted along each line, (columns 8 and 9) averages calculated from these counts and reduced to the standard magnification of 85 diameters and the standard linear unit of 1 in., (column 10) ratios of the number of grains in the direction of rolling to the number in the reverse direction, and (column 11) the calculated number of grains per square inch at the standard magnification. To obtain the actual number of grains per square inch of etched surface, the latter value should be divided by 85×85.

The ratios of the number of grains in the direction of rolling to the number in the reverse direction are not far from unity in the case of every anneal from which counts were obtained. This indicates that the grains in the annealed structures are equiaxed, which would naturally be expected in the event of complete recrystallization. In the case of the 450° anneal, this ratio, 0.87, is far enough below unity to give rather positive indication that the axial relationships developed in rolling have not been completely removed by anneal at this low temperature, *i.e.*, that the material has not completely recrystallized.

On the basis of the counts, we can distinguish with ease between the four anneals at 100° intervals when suitable magnifications are used in counting. After recalculating the results in terms of number of grains per square inch, the differences between the 450° and 550° anneals, and the 550° and 650° anneals are more marked than between the 650° and 750° anneals. In other words, the number of grains per square inch increases rapidly in the early stages, but more slowly as the temperature increases. From a practical standpoint, one would be inclined to divide the entire range into four stages; one in the neighborhood of 350° giving partial recrystallization with barely distinguishable grains (at low magni-

[17] G. H. Gulliver: *Metallic Alloys*, Chapter VII, p. 228 (London, 1913).

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fication), another in the neighborhood of 450° giving a count ranging from, say, 300 to 400 grains per square inch at 85 diameters, another in the neighborhood of 550° giving a count ranging from 50 to 100, and another in the neighborhood of 700° giving a count ranging from 5 to 10 grains. Other stages would be recognized as intermediate. This procedure, with general observations relative to homogeneity, etc., would suffice to characterize the annealing treatment.

In connection with the above statements, it may be added that some latitude must be allowed in interpreting the count, since there are counting errors amounting to at least 10 per cent. of the value obtained where the troublesome twinned grains occur in quantity and, aside from all difficulties associated with one and the same material, a second batch of metal, close to the first in composition and treated as far as possible in the same way may give somewhat different results. The following comparison will illustrate this:

The present lot of metal containing 66.65 per cent. copper, 0.30 per cent. lead, and 0.08 per cent. iron, rolled to 50 per cent. reduction after anneal at about 650° in the mill gave a count of 95.9 grains per square inch at 85 diameters and a tensile strength of 49,025 lb. per square inch after a 30-min. period of anneal at 550°.

Another lot of metal containing 66.56 per cent. copper, 0.26 per cent. lead, and 0.08 per cent. iron, rolled to 50 per cent. reduction after anneal somewhat above 650° in the mill (estimated) gave a count of 67.2 grains per square inch at 85 diameters and a tensile strength of 47,100 lb. per square inch after a 30-min. period of anneal at 550°.

There is a difference of about 30 per cent. in the number of grains per square inch and of about 4 per cent. in the tensile strength. It appears that, with due allowance for errors in counting, this method is more sensitive in detecting variations in strength properties than the tensile tests themselves.

With regard to the influence of the previous degree of deformation on the properties after anneal, both Rose[1] and Grard[2] showed that the properties investigated, viz., scleroscopic hardness in the first case and tensile strength, elongation, etc., in the second, were independent of the degree of (severe) deformation, provided the temperature of anneal was above a certain (low) value. In an earlier section (p. 633) we have shown that structural identity results from anneal at 650°, in the case of cartridge brass, when the previous reduction by rolling was not less than 20 per cent. From a practical standpoint, this amount of reduction will give about the same properties after anneal at 550° as a reduction of 50 per cent., although appreciable differences in the size of grain are encountered. Table VI illustrates this statement.

[1] *Journal of the Institute of Metals*, vol. viii, pp. 86–125 (1912).

[2] *Revue d'Artillerie*, Feb.–Apr. (1909).

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TABLE VI.—*Summary of Properties after Half-Hour Anneal at 550°—Brass Containing 66.56 Per Cent. Copper, 0.26 Per Cent. Lead, 0.08 Per Cent. Iron*

Initial Reduction, Per Cent.	Elongation in 1 in., Per Cent.	Reduction of Area, Per Cent.	Tensile Strength, Pounds Per Square Inch	Scleroscope	Grains per Square Inch at 85 Diameters
20	72	59.5	46,739	10	55.5
50	73	62.0	47,100	10	67.2

Here again, the grain count furnishes the best means of distinction between two similar kinds of material.

Turning now to Fig. 4, in which the physical properties and the number of grains per unit area in the present series are shown as functions of the annealing temperature, we note that the inflexional characteristics of the curves which represent physical properties are not consistently traceable to similar characteristics affecting the curve of grain counts. In other words, the grain size is not the only factor which determines the value of any given property. Some points of similarity may, however, be observed. Thus, the curves representing percentage elongation rise rapidly in the temperature region which corresponds to a rapid fall on the grain-count curve and reach maxima in the region which corresponds to the most gradual changes in grain size. The curves representing hardness and tensile strength bear no close relation to the grain-count curve, although there is a slight falling off in the temperature-rate of change toward the end of these curves which corresponds to a marked falling off in the temperature-rate of change on the grain-count curve. On the other hand, the curve representing reduction of area of section in the tensile test shows a maximum in the temperature region which corresponds to rapid change along the grain-count curve. The maximum reduction of area almost invariably occurs at a lower temperature than the maximum elongation, although the relative positions of these points, as well as the character of the maxima (whether sharp or flat) are generally influenced by both soluble and insoluble impurities. The highest combined values of reduction of area and elongation are obtained in this mixture by anneal at a temperature in the neighborhood of 650°. This corresponds to a count of about 8 grains per square inch at 85 diameters, which value may be taken as the controlling factor in developing these properties.

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FIG. A.—From sample which had softened 2½ points after 10 min. at 325°C. × 1,000.

FIG. B.—Softening of 1 point with rise of temperature to 325°C. × 1,000.

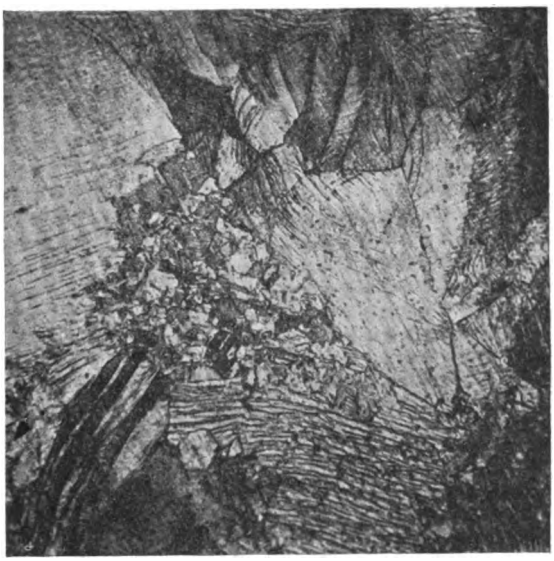

FIG. C.—Softening of 4 points after 30 min. at 300 to 325°C. × 500.

FIG. D.—Softening of 1 point as in Fig. B. × 1,000.

PLATE I.—EARLY STAGES OF RECRYSTALLIZATION IN CARTRIDGE BRASS (COPPER, 69.35; IRON, 0.04; LEAD, 0.03 PER CENT.) AFTER 40 PER CENT. REDUCTION BY COLD ROLLING. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE

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FIG. A.—5 per cent. reduction. Gage 18½.

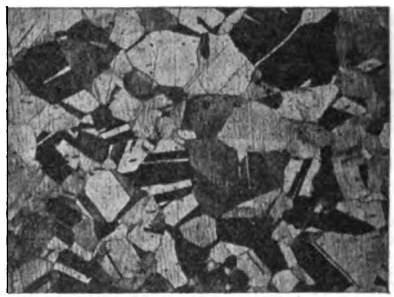

FIG. B.—10 per cent. reduction. Gage 19.

FIG. C.—20 per cent. reduction. Gage 20.

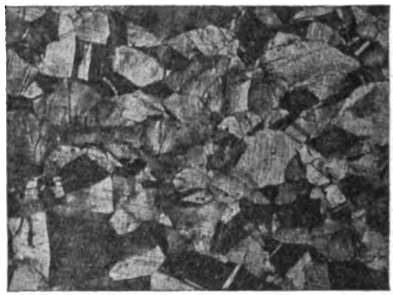

FIG. D.—29 per cent. reduction. Gage 21.

FIG. E.—37 per cent. reduction. Gage 22

FIG. F.—43.5 per cent. reduction. Gage 23.

FIG. G.—50 per cent. reduction. Gage 24.

FIG. H.—55 per cent. reduction. Gage 25.

PLATE II.—COMMON HIGH BRASS REDUCED IN STAGES FROM GAGE 18 to GAGE 25 AFTER SOFT ANNEAL IN THE MILL. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 72.

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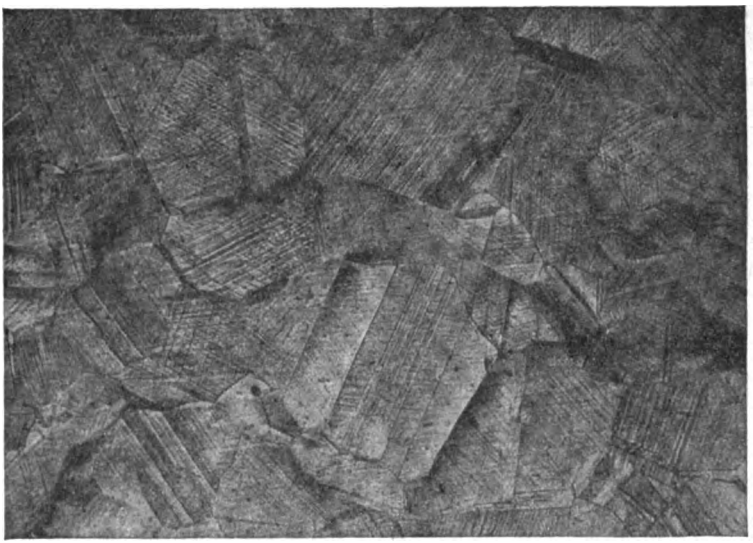

FIG. A.—Strained after anneal at 750° following 50 per cent. reduction.

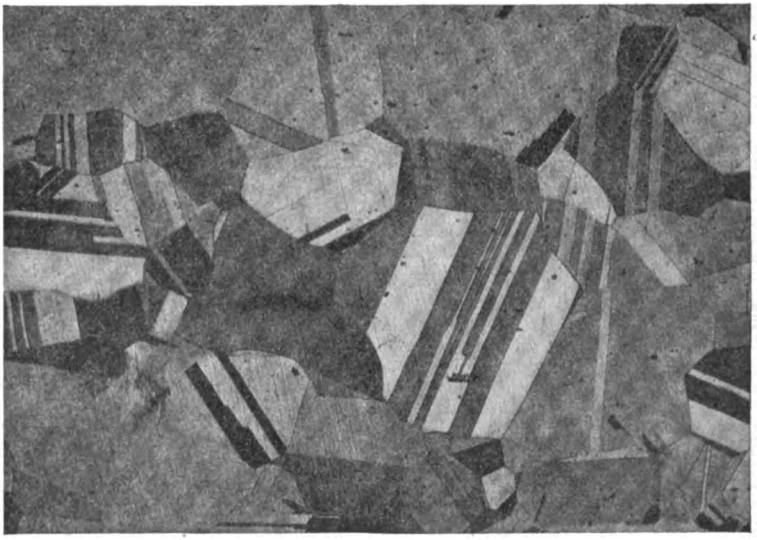

FIG. B.—Annealed 15 min. at 615° following treatment given above.

PLATE III.—ADMIRALTY MIXTURE (COPPER, 72.53; IRON, 0.07; LEAD, 0.04; TIN, 1.06 PER CENT.). ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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FIG. A.—Strained after anneal at 750°.

FIG. B.—Annealed 15 min. at 615° following treatment of Fig. A.

FIG. C.—Strained after anneal at 775°.

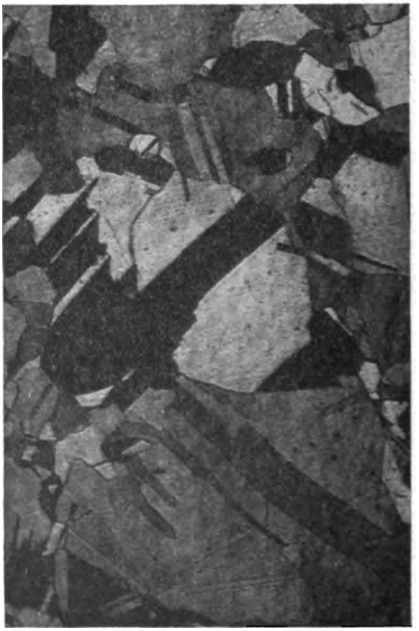

FIG. D.—Annealed 20 min. at 550° following treatment of Fig. C.

PLATE IV.—ADMIRALTY MIXTURE (COPPER, 72.53; IRON, 0.07; LEAD, 0.04; TIN, 1.06 PER CENT.). ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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FIG. A.—2 per cent. reduction.

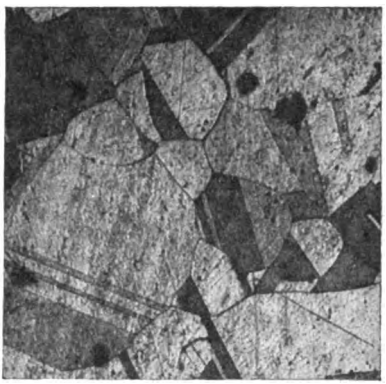

FIG. B.—30 min. at 350°C.

FIG. C.—30 min. at 450°C.

FIG. D.—30 min. at 550°C.

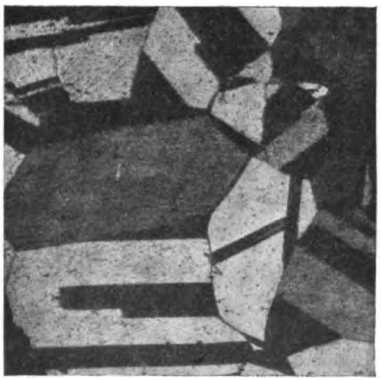

FIG. E.—30 min. at 650°C.

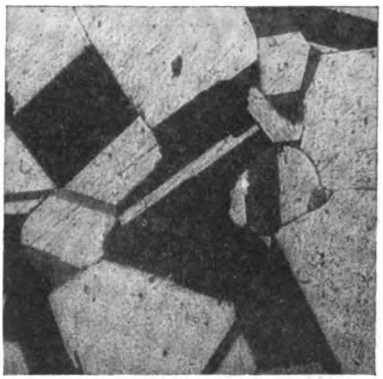

FIG. F.—30 min. at 750°C.

PLATE V.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. EFFECT OF ANNEAL AFTER 2 PER CENT. REDUCTION BY ROLLING. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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FIG. A.—4 per cent. reduction.

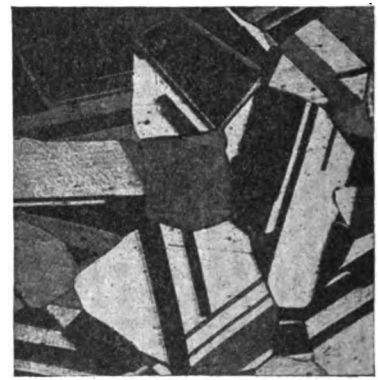

FIG. B.—30 min. at 350°C.

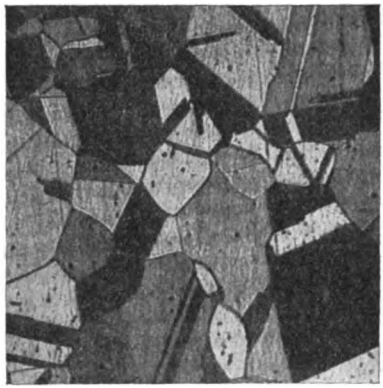

FIG. C.—30 min. at 450°C.

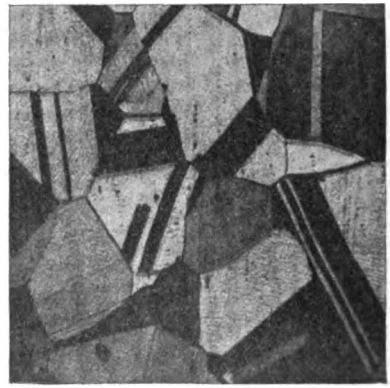

FIG. D.—30 min. at 550°C.

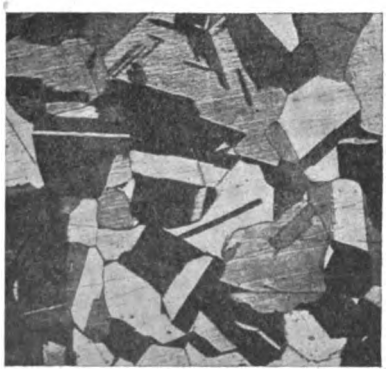

FIG. E.—30 min. at 650°C.

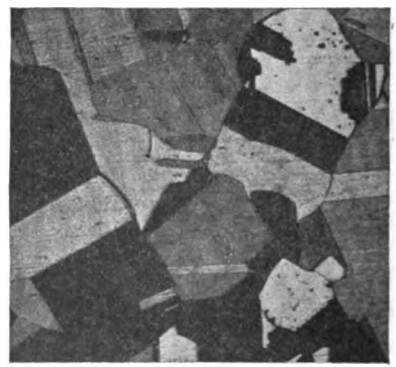

FIG. F.—30 min. at 750°C.

PLATE VI.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 4 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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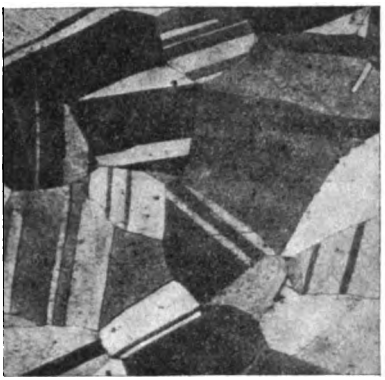

FIG. A.—8 per cent. reduction.

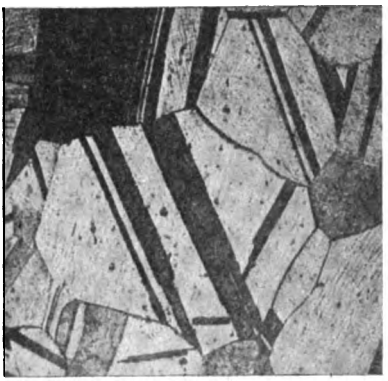

FIG. B.—30 min. at 350°C.

FIG. C.—30 min. at 450°C.

FIG. D.—30 min. at 550°C.

FIG. E.—30 min. at 650°C.

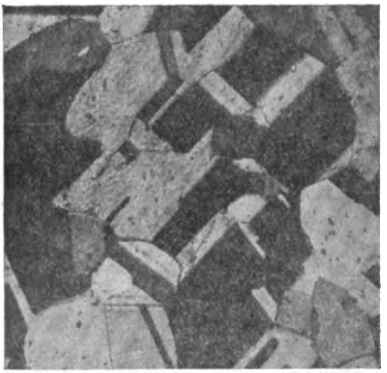

FIG. F.—30 min. at 750°C.

PLATE VII.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 8 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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FIG. A.—12 per cent. reduction.

FIG. B.—30 min. at 350°C.

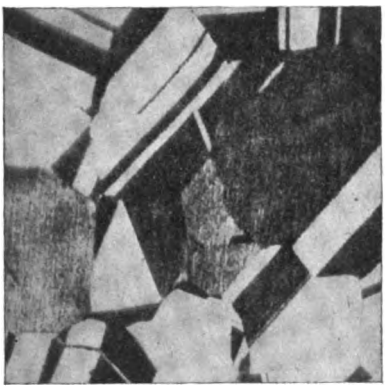

FIG. C.—30 min. at 450°C.

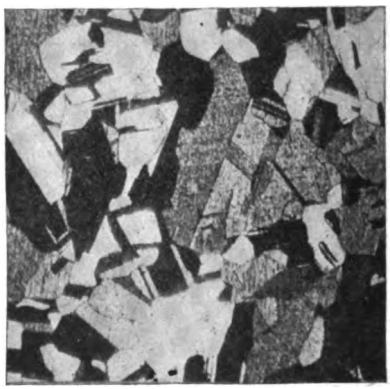

FIG. D.—30 min. at 550°C.

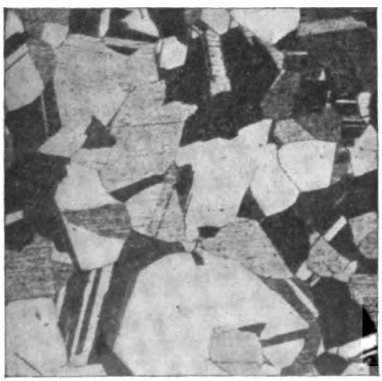

FIG. E.—30 min. at 650°C.

FIG. F.—30 min. at 750°C.

PLATE VIII.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 12 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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FIG. A.—15 per cent. reduction.

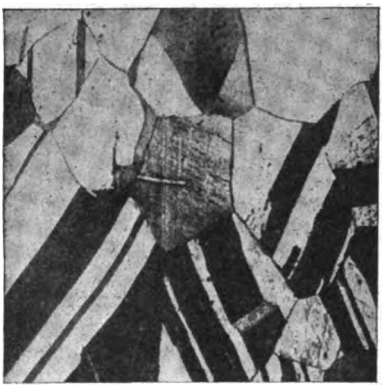

FIG. B.—30 min. at 350°C.

FIG. C.—30 min. at 450°C.

FIG. D.—30 min. at 550°C.

FIG. E.—30 min. at 650°C.

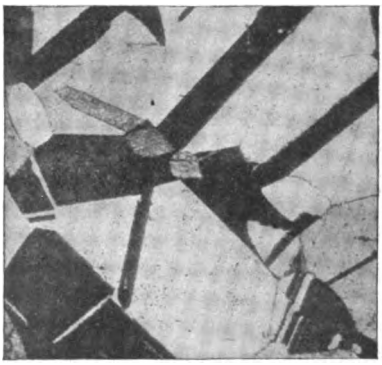

FIG. F.—30 min. at 750°C.

PLATE IX.—CARTRIDGE METAL (COPPER 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 15 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. AT 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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FIG. A.—25 per cent. reduction.

FIG. B.—30 min. at 350°C.

FIG. C.—30 min. at 450°C.

FIG. D.—30 min. at 550°C.

FIG. E.—30 min. at 650°C.

FIG. F.—30 min. at 750°C.

PLATE X.—CARTRIDGE METAL (COPPER, 70.16; IRON, 0.06 PER CENT.; LEAD, TRACE). EFFECT OF ANNEAL AFTER 25 PER CENT REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED 20 MIN. at 800°C. BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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FIG. A.—50 per cent. reduction.

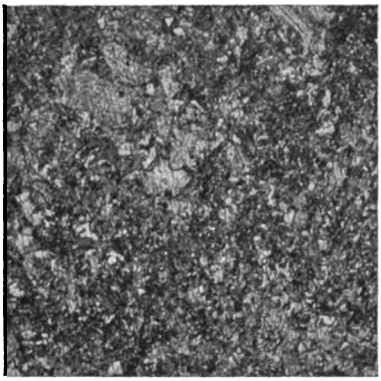

FIG. B.—30 min. at 350°C.

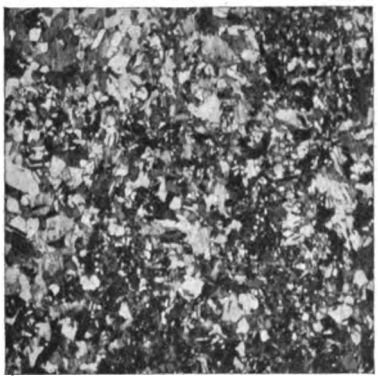

FIG. C.—30 min. at 450°C.

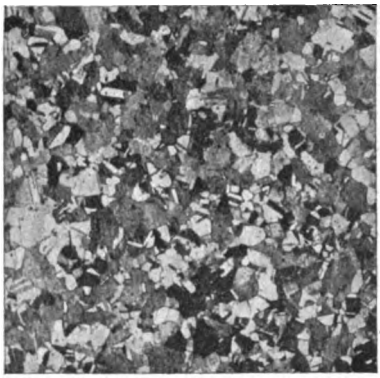

FIG. D.—30 min. at 550°C.

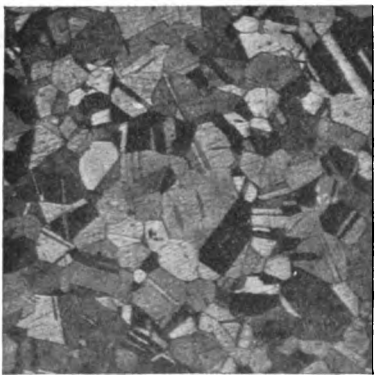

FIG. E.—30 min. at 650°C.

FIG. F.—30 min. at 750°C.

PLATE XI.—ALPHA BRASS (COPPER, 66.65; IRON, 0.08; LEAD, 0.30 PER CENT.). EFFECT OF ANNEAL AFTER 50 PER CENT. REDUCTION BY ROLLING. ORIGINAL STRIP ANNEALED SOFT IN THE MILL BEFORE REDUCTION. ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 85.

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Discussion

Zay Jeffries, Cleveland, Ohio (communication to the Secretary*).—The authors are to be congratulated for this interesting and valuable contribution. The paper as a whole shows very careful experimentation coupled with rare judgment in drawing conclusions.

I have been working along similar lines for the last two years with other metals and alloys. In a short conversation with Dr. Mathewson about July 1, 1915, he mentioned the fact that different degrees of cold work greatly affect the temperature of recrystallization on annealing. Later I determined the recrystallization temperatures of iron, mild steel, and copper, with different degrees of cold work and found that the greater the degree of cold work, the lower the temperature of recrystallization on annealing. In the course of these experiments, I conceived practically the same ideas regarding the reasons for recrystallization as those advanced in the article under discussion.

After a metal has been reduced by cold work to about one-fourth of its original area, the differences in orientation of different parts of a given grain can be seen plainly at a magnification of about 1,000 diameters.

It is interesting to note that in Fig. A, Plate I, the lines of deformation are not parallel. For example, in the large grain at the lower left corner of the micrograph, if the lines of deformation about ⅜ in. from the left edge are continued in a straight line, this line will intersect a straight line drawn as a continuation of the lines of deformation on the extreme right of the grain, at an angle of about 20°. Similarly, if the lines of deformation at the extremities of the large grain in the right central portion of the micrograph are extended, the lines intersect at an angle of about 35°. The individual lines of deformation are not straight. In some of these the parts of the broken lines intersect at angles as great as 15°.

This is positive evidence of change of orientation of different parts of the same grain as a result of cold work. However, the micrograph permits us to study only one plane. If we consider the plane of the micrograph as one of the three standard coördinate planes, then we must consider that there is change of orientation within each grain with respect to the other two planes. While there is generally a maximum change in orientation with respect to one of the planes, depending on the original specific orientation of the grain and the manner and direction in which the deformational forces were applied, there is some change in orientation with respect to each plane. Such a destruction of the original crystalline arrangement would naturally be more complete as the degree of cold work increased.

* Received Jan. 28, 1916.

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It seems logical to take the stand that if differences in orientation can be seen within the same grain, after cold work, it is proof that such differences exist; and that if differences cannot be seen by our ordinary microscopic methods, it is no proof that such differences do not exist.

In defense of the latter contention, it has been my experience that etching pits are the only real guide to specific orientation and it is possible to have differences in orientation within the same original grain and yet have the etching pits appear to be in the same orientation. There might be changes in the relative dimensions of the etching pits, but owing to the selective action of the etching reagent, these relative dimensions may vary with the same orientations. Furthermore, by methods known at present, the specific orientation as shown by the etching pits can only be roughly determined.

If we consider the quantity of light reflected by the various grains to the eyepiece of a microscope (using say vertical illumination) as a measure of orientation, we err greatly. As an example, let us suppose one of the grains which appears the lightest in color to have the flat surfaces of the crystal units parallel with the polished plane. The crystal units could be revolved 360° in the plane of the polished section without changing the quantity of reflected light in any of the infinite positions. On the other hand, suppose one of the darkest-colored grains has the diagonals of the crystal units perpendicular to the plane of the polished section. These crystal units, having their corners pointing toward the source of illumination, will reflect the light outside the microscope tube and appear dark. They can be revolved 360° in the plane of the polished section and the grain will appear dark in every one of the positions. These crystal units can also be revolved through certain appreciable angles in any other plane without causing appreciable change in the appearance to the eye. This statement can be verified readily by the macroscopic examination of a deeply etched, coarse-grained section, using oblique illumination. It is apparent, therefore, that differences in orientation within a grain after cold work may be sufficient to cause recrystallization on heating and yet escape the observer's notice in the ordinary microscopic examination.

It seemed to me that Bragg's x-ray spectrometer might be used advantageously in determining specific orientations, as well as in studying the changes in orientation during the recrystallization period. In correspondence conducted through James G. Biddle, Philadelphia, Mr. Bragg said that no such work had been attempted with the x-ray spectrometer but he thought it likely that positive results could be obtained. I have not carried the matter any further, to date.

On p. 626, the authors make the following statement: "This diagram (Fig. 2) is based upon the assumption that the mean size of the new grains which form at any given temperature by coalescence of existing

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grains or by local breakdown of internal surfaces, within the original strain-hardened grains, is determined by the temperature of anneal."

I will take this opportunity to announce certain results bearing on the above which I have obtained during the past two years. The actual tabulated data, together with micrographs and more detailed explanations, will appear later in a separate paper.

The conclusions which follow, are based on experiments with cold-worked wires of various metals, including iron, mild steel, copper, aluminum, and nickel. The degree of cold work includes the ranges between 15 per cent. and 99.99 per cent. reduction in area.

There is some question as to whether a grain-size equilibrium is reached at a certain temperature after a certain comparatively short period of time, but there is no doubt that in reheated cold-worked metals, the grains grow rapidly up to a certain point, after which the change in grain size, if there be any, is extremely slow. Hereafter, where the term "grain-size equilibrium" is used, it will be understood to mean the point where grain size either changes none or slowly. The authors contend that the equilibrium grain size depends on the temperature of anneal. The term as used by them is defined as that grain size at a certain temperature, which will not change further, irrespective of additional time, at that temperature. It also implies the condition whereby a slight increase in temperature will cause grain growth, resulting in a new equilibrium grain size for the new temperature. The latter condition is not considered when I use the term "equilibrium grain size" since it has been found possible to obtain a grain size at a certain relatively low temperature, which does not change with increase in temperature even to the fusion point of the metal or alloy.

The following general rules apply to pure metals, solid solutions, and alloys consisting largely of a pure component or a solid solution, cold worked by reduction of area 15 per cent. or more. Regions in which allotropic changes occur are expected.

1. The grain-size equilibrium at any temperature above the equiaxing range depends largely upon the *rate* at which the metal is heated to that temperature, and more particularly on the rate at which the equiaxing range is passed.

(*a*) The faster the heating rate, the smaller the equilibrium grain size.

(*b*) The slower the heating rate, the greater the equilibrium grain size.

As an extreme example of the above, small wires have been heated to within 100° C. of the melting point, by means of electric current. When the current is started low and is increased gradually the equilibrium grain size is sometimes 200 times as great as when the necessary current is applied instantaneously.

2. To produce the smallest possible equilibrium grain size the metal

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should be heated quickly to a temperature just above that of equiaxing. An increase in temperature will, of course, cause further grain growth.

Due to the mechanical difficulties in heating large pieces of metal quickly, it is possible to produce a smaller equilibrium grain size in a small section than in a large section of a metal.

3. To produce the coarse-grained structures, the metal should be held in the equiaxing range for an extended period of time. In some cases a grain size develops which does not change on further heating to the fusion point. Generally, the final coarse grain can be obtained in a shorter time by holding in the equiaxing range for a certain time and then slowly raising the temperature to near the melting point of the metal.

In some cases when a small wire is held at the equiaxing temperature for only a few seconds, it develops coarse grains which do not change in size on further heating.

The history of the growth of the grains in the equiaxing range has been studied. It is obvious that if the samples are heated by electric current, the center will be the hottest portion and there will be a gradual decrease in temperature toward the surface. If just sufficient current is used, so that the central portion of a cross-section of the sample (assuming a sample of considerable length) reaches the equiaxing temperature, then the outside portions will not equiaxe. A sample treated in this way for a certain time depending on the metal or alloy and on the degree of cold work will first develop the coarser grains at the center of any cross-section. After a time, these grains invade the unequiaxed portions and grow to the surface of the sample on all sides, producing a radial grain structure similar to that of a slowly cooled, cast metal.

By increasing the amperage so that about half the area of the wire is above and half below the mean equiaxing temperature, the coarse grains first form on the circumference of the circle dividing these regions. As time increases, these large grains grow both toward the center and toward the surface of the wire, giving a final grain structure similar to that of the preceding sample.

When the amperage is raised slightly so that only a small area near the surface remains unequiaxed, the coarse grains form at the boundary between the equiaxed and unequiaxed portions and grow to the surface and also to the center of the sample. The final grain structure is so like the other two examples cited that unless examinations had been made at several intervals on each sample, one might think they were formed in the same way.

If the amperage is increased so that the whole cross-section of the metal is above the equiaxing temperature, a fine-grained, rather uniform structure results.

It is evident that the coarse grains first formed in each case at points of equal temperature. This temperature, I have called the "Critical

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Temperature for Grain Growth." It is defined more particularly as the minimum temperature at which two adjacent grains can coalesce to form one larger grain, provided that this larger grain will have sufficiently increased its power of attack to enable it to absorb adjacent grains which cannot coalesce with each other. Time is always to be understood as a factor governing the first stages of grain growth.

The critical temperature for grain growth is different for different metals and for different degrees of cold work in the same metal as the authors have so conclusively shown. If their conclusions regarding recrystallization are correct, the critical temperature for grain growth should be the same as the equiaxing temperature. Certainly the two cannot be far from the same temperature, but it should be noted that the development of the coarse grains is a phenomenon of grain growth and not one of equiaxing. It should be further noted that the portions of the metal which are heated above the equiaxing temperature do not develop the coarse-grained structure.

Let us consider the conditions which exist just above and just below the critical temperature for grain growth, as for example, in a wire heated by electric current. In the portion which is at or slightly above the critical temperature for grain growth, two adjacent grains will coalesce to form a larger one, but each of these grains was adjacent to another grain on the colder side. They coalesced with each other in preference to their colder neighbors, because their plasticity was greater, due to higher temperature, thus making the resistance to coalescence lower. Each grain has a certain power of "attack" (power to force other grains into its own orientation or at least into one which is more like that of the attacking than the absorbed grain) which is greater the larger the grain. In accordance with equilibrium laws, the colder grains will coalesce with an infinitesimal increment in the force of grain growth or decrement in resistance to grain growth. By the coalescence of the two first grains, both of these conditions would be brought about. The increased size of the new grain would give it enough greater power of attack to cause absorption of these cooler grains, and, if the amorphous cement theory is true, some amorphous metal will have been changed to crystalline and by liberating its heat of crystallization the temperature will be slightly increased locally.

It might be argued that the amount of heat consumed in changing the orientation would be more than equal to the amount liberated by the crystallization of the amorphous material. However, the increased size of the grain is probably the controlling factor in the power of attack. It is the relative sizes of adjacent grains, other things being equal, that determines the ability to coalesce. While the power of attack has been increased in the coalesced grains, the power of defense has remained unchanged in the colder grains.

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With the four original grains now as one, the power of attack is so increased that other adjacent small grains are easily absorbed. The grains in the region below the critical temperature for grain growth are not able to coalesce with one another and, therefore, remain small and have a correspondingly small power of defense. They are easily absorbed by the grains which are several times larger. These large grains even invade the unequiaxed portions and eventually grow to the surface of the sample.

What has happened in the meantime in the portion above the critical temperature for grain growth? Since it was possible for grain growth to take place, in nearly all parts of this region, grains would begin to coalesce simultaneously at a great many centers. When the region dominated by each center became one grain, approximate grain-size equilibrium would result. This portion would then consist of a rather uniform, fine-grained structure.

When the grains which develop at the critical temperature for grain growth reach sizes considerably larger than the average grain size in the hotter portion, the latter grains are readily absorbed and the coarse-grained structure is thus extended to the center of the sample.

Heating a sample by electric current might be considered an ideal condition for bringing about this selective grain growth since it affords a means of obtaining large temperature gradients from the center of the sample to the surface; also it is an ideal method for preventing the selective grain growth, that is, producing a fine-grained structure, since the sample can be heated past the critical temperature for grain growth probably quicker this way than any other.

By furnace heating, it will often be found difficult to obtain the very coarse grains, especially if the sample has been cold worked by reducing its area more than about 40 per cent. or less than about 10 per cent. When a piece of metal is cold worked to a certain average amount, it will be found that all parts of a section are not deformed in the same degree. The portions near the outside where the force was applied will receive more cold work than the center. As the outside increases in hardness, the deformation is forced upon the softer center, but there is always a gradient of degree of cold work, which is greater the less the total amount of cold work. There are also local differences in degree of cold work due to the size of the grains and their directional properties as the authors have pointed out. This gradient in degree of cold work will produce a gradient in temperature of equiaxing. Since in furnace heating the temperature difference between the surface and center of a piece of metal is slight, it is not difficult to maintain one portion of a moderately cold-worked sample at the critical temperature for grain growth with other portions above and below and thus cause selective grain growth.

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With excessive cold work, the surface and central portions approach the same equiaxing temperature, a fact which makes the necessary conditions for coarse-grained formation hard to obtain with furnace heating. However, there is usually a noticeable degree of selective grain growth even with excessive cold work and furnace heating when the proper temperature is used and sufficient time is given. With electric-current heating, this coarse-grained structure has been produced in a wire that had been cold-worked by reduction in area of 99.99 per cent. without intermediate annealing.

The metals which equiaxe at low temperatures sometimes require hours to form these coarse grains at the critical temperature for grain

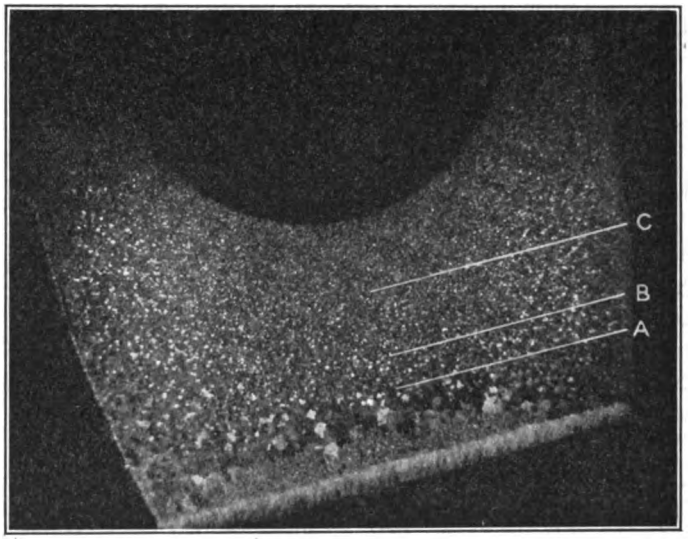

FIG. 1.

Steel. Carbon 0.05 per cent. Magnified 5 diameters. Subjected to Brinell ball test under pressure of 6,000 kg. and heated to 650° C. for 7 hr. Vertical section through bottom of spherical depression. (Fig. 27 from Sauveur's *Metallography of Iron and Steel.*)

growth and metals having high equiaxing temperatures usually require less time.

By application of this knowledge, it is possible to explain a great many so-called mysteries concerning grain growth. One which I will attempt to explain is that of the "critical strain" observed by Professor Sauveur.[1] Professor Sauveur has kindly given permission to have reprinted here the micrographs shown in Figs. 1 and 2 which are Figs. 27 and 30, respectively, in *The Metallography of Iron and Steel.*

It is apparent that the degree of cold work in each of these samples

[1] The International Association for Testing Materials, Sixth Congress, New York, 1912. Also *The Metallography of Iron and Steel*, 1912.

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varies and that the equiaxing temperatures will vary inversely. The temperature used was in each case 650° C. and the time, 7 hr. The equiaxing temperatures of the portions which were severely cold worked were less than 650° C. and those of the milder cold-worked portions were greater than 650° C. The boundary between these two portions would be at the critical temperature for grain growth and would develop coarse grains. The lower equiaxing regions would show a progressively finer grain as the equiaxing temperature decreased, due in part to the more even local equiaxing temperature and in part to the faster rate of heating through the equiaxing range, the latter being helped by the lesser plasticity of the metal at the lower temperature, resulting in a decreased velocity of grain growth at the temperatures where selective grain growth was possible.

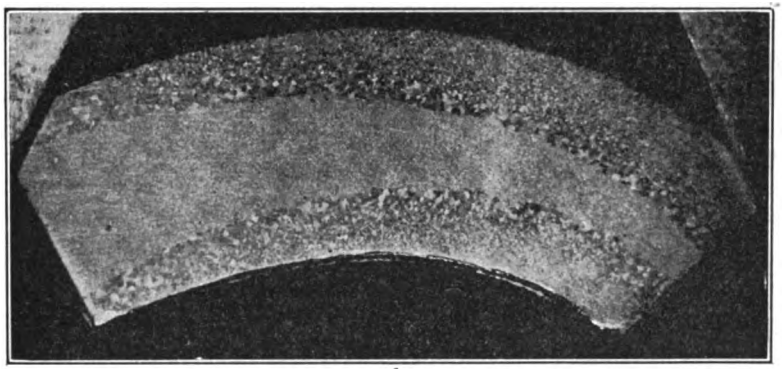

FIG. 2.

Steel. Carbon 0.05 per cent. Magnified 2.5 diameters. Bar bent double and heated to 650° C. for 7 hr. Longitudinal section through center of bent portion. (Fig. 30 from Sauveur's *Metallography of Iron and Steel.*)

The portions having their equiaxing temperatures above 650° C. would not change in grain size except by invasion of the coarse grains. It is probable that if sufficient time were given, these coarse grains would completely conquer the unrecrystallized portions. The fact that they were invading the unrecrystallized areas when removed from the furnace is clearly shown by the distinct line of demarcation between these and the coarse-grained portions.

It is thus seen that the phenomenon ascribed to a critical strain is really due to a critical temperature for grain growth. It is true that this critical temperature changes with the strain, but since I have found that the laws of critical temperature for grain growth hold equally well for compressed metals (metals pressed into briquets from their powders and heated in order to sinter and make crystalline without fusion) in which strain plays practically no part, it is evident that critical strain would not be a sufficiently broad term to cover all such cases. Further-

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more, similar results can be obtained with any degree of strain above about 15 per cent. reduction in area by cold work, if the proper temperature conditions are used.

Now, let us examine the authors' micrographs with the above ideas in mind.

In Plates IX, X, and XI, all of the samples which were heated to 550° C. and above are completely recrystallized. The anneals at 550° C. in these plates show a progressively decreasing grain size as the equiaxing temperature decreases. The same is true of the 650° C. and the 750° C. anneals. In the latter case, the measurements from the micrographs show grains as follows: Fig. F, Plate IX, 16 grains; Fig. F, Plate X, 25 grains; Fig. F, Plate XI, 41 grains.

The time consumed in bringing these samples up to 750° C. was the same. The samples having the greatest degree of cold work will have low equiaxing temperatures and comparatively short ranges between the beginning and the end of equiaxing. Owing to the method of heating, the rate would be faster at the lower temperatures. These conditions would defeat selective grain growth in the low equiaxing metals and a finer grain at a given temperature above the equiaxing range would be expected.

It should be noted that the difference in grain size between the corresponding micrographs in Plates X and XI is greater than between those in Plates IX and X. These differences check qualitatively with the differences in degree of cold work which are as follows: Plate IX: Plate X: Plate XI: 15:23:50, respectively. In fact, these samples might be considered as duplicates in brass of certain portions of Fig. 27. If we consider the samples annealed at 550° C., Fig. D, Plate IX, would be represented by a section at *A*, Fig. D, Plate X, at *B*, and Fig. D, Plate XI, at *C*.

It would be interesting to hold the temperature in the equiaxing range in these brass samples and then heat to various higher temperatures. For example, in Fig. C, Plate IX, are the large grains recrystallized and in the process of absorbing the smaller ones, or are they in the original cold-worked state? The scleroscope results would indicate the latter, but the number of these large grains might be small and have little effect on the average of a number of readings. At any rate, if this sample after receiving its treatment for 30 min. at 450° C. were heated to 550° C. and held there for 30 min. longer, the grain size should be decidedly greater than Fig. D, Plate IX, even though the latter were heated an additional 30 min. to make the total time the same in each case.

C. H. MATHEWSON, New Haven, Conn. (communication to the Secretary*).—I have taken great pleasure in reading Mr. Jeffries' discussion

* Received Mar. 24, 1916.

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of the recrystallization phenomena described in this paper and that previously described by Professor Sauveur, in the light of his own (Mr. Jeffries') observations in this field. Mr. Jeffries has made an important contribution to the general subject of recrystallization after cold working in showing that growth of grain on annealing is particularly pronounced in the temperature region in which an annealing effect first occurs at the given value of deformation. If such growth is as pronounced in the case of moderately or severely worked alpha brass as Mr. Jeffries seems to have observed in other cases, or as appears in the illustrations from Professor Sauveur's paper, it would have an important bearing on the commercial annealing of brass, in that it would render the load factor of great importance with respect to the ultimate degree of grain homogeneity secured, *i.e.*, a heavy load of metal would pass more slowly through the temperature region in question than a light load and would tend to develop local coarsening of grain.

Although Mr. Phillips and I did not make experiments in this particular direction, many detached observations make it difficult to believe that in the ordinary muffle annealing of rather severely worked flat metal, for example, where the deformational gradient is reasonably flat and the temperature gradient likewise flat, the grain size will be uniform and essentially characterized by the finishing temperature of anneal as ordinarily chosen (say, in the neighborhood of 600° C., within wide limits of loading (rate of heating). This observation is not intended as a refutation of the ideas advanced by Mr. Jeffries, which I am fully prepared to believe. It is merely an expression of opinion as to the general magnitude of the effect in question under conditions originally dealt with in the paper.

From a similar viewpoint, a few words may be added relative to Mr. Jeffries' interpretation of our micrographs from metal annealed at the same temperature after different degrees of reduction by cold-work. He points to a decrease in grain size with increasing degree of deformation which is quite apparent in any of the series of micrographs cited, and this, we are prepared to believe, is partly due to the rate effect in question. But it must be borne in mind that, in every case, we started with very coarse-grained metal (annealed at 800° before reduction) and, in the 550° C. set of anneals, for example, the principal cause for the striking decrease in visible grain size with the degree of deformation (Figs. D of Plates VIII, IX, and X) is the decreasing size of unrecrystallized fragments of the original grain, which, of course, are not grains which have grown in these tests and do not represent selective growth of grain, but progressive recrystallization of original grain substance.

Disregarding the scleroscopic evidence in answering Mr. Jeffries' query whether the large grains in Fig. C, Plate IX, are "recrystallized and in the process of absorbing the smaller ones, or are in the original

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cold-worked state," the very apparent curvature of some of the twin-bands (lower, right-hand corner) indicates that they are in the cold-worked state. The irregular boundaries separating the fragments from the recrystallized conglomerate referred to in the paper in connection with Fig. C, Plate I, is here very apparent. Finally, a subsequent anneal of 30 min. at the next higher temperature (suggested by Mr. Jeffries) shows a breakdown, instead of an increase in size of the large grains, which is, of course, conclusive. The accompanying micrograph (Fig. 3) shows the ordinary anneal at 550° C. and the re-anneal at 550° C. (of the material previously annealed at 450° C.) side by side. Both specimens show considerable variations in grain size from point to point, owing to the presence of unrecrystallized fragments (smaller than in the 450° C.

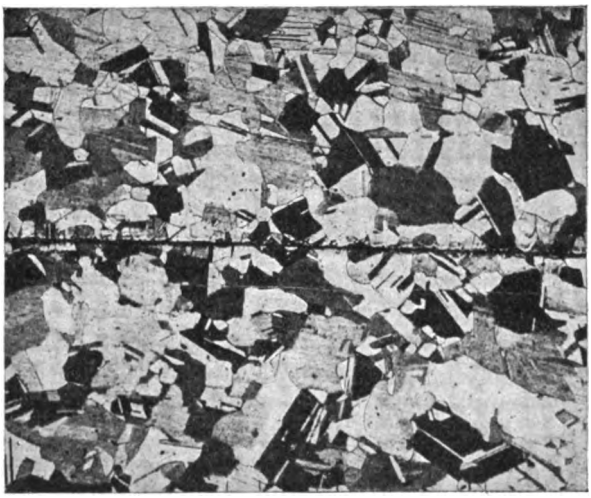

FIG 3.—Top: Reduced 15 per cent. by Rolling. Annealed 30 min. at 450° C. and then 30 min. at 550° C. × 73.
Bottom: Reduced 15 per cent. by Rolling. Annealed 30 min. at 550° C. × 73.

anneal) and of recrystallized grains side by side. It would not be logical to make a comparison by count on any single spots as magnified, but an examination of the whole surface in both cases seems to indicate that the portions shown offer a fair general comparison.

Unrecrystallized fragments need not be considered in the case of anneals at 750° (Figs. F, Plates IX to XI), and it seems necessary to attribute any progressive decrease in the size of grain, such as Mr. Jeffries observes, to the rate effect which he has described, except that Fig. F, Plate XI should not rightfully be included in any series of counts, as it refers to a mixture of somewhat different composition and previous treatment.

In presenting the diagram of fragmentation and grain growth (p.

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627 of the paper), the authors were not unaware of the difficulties in the way of assuming without modification that a definite size of recrystallized grain corresponds to a definite annealing temperature. Such an assumption is, however, very attractive and need not be entirely rejected on account of the apparent contradictions which lie in the phenomena of selective grain growth by anneal after local deformation or through a deformational gradient, as described by Robin[1] and by Sauveur, and interpreted by Jeffries in the present discussion. The broad question affects the coördinate relationships between the curves of fragmentation (distribution of fragments according to size at a given degree of deformation) and the respective curves of grain-size equilibrium (temperature-grain size); in particular it requires the introduction of an assumption that the true equilibrium grain size is not reached in many cases even after long periods of anneal, as in Professor Sauveur's Fig. 27, but that growth of grain is subject to a special impetus during sojourn in the temperature region of initial softening for a given deformation (at the equiaxing temperature, as expressed by Jeffries, or after critical strain, implying the use of an associated temperature, as expressed by Sauveur), carrying the grain nearer the true equilibrium size than even a much longer anneal when the initial softening temperature is passeed quickly, as in the case of the neighboring, more severely strained metal. The latter remains in its temperature region of initial softening only during a fraction of the preheating period and to imagine, with Jeffries, that simultaneous coalescence at numerous points (which must, of course, occur in a region above the initial temperature range of softening) would result in a low order of grain size is not unlike imagining with Carpenter that the alpha and gamma phases, which occur in apparent beta brass below the eutectoid temperature, persistently remain very fine-grained because they have separated simultaneously at countless points. In the case of the two-phase system, however, we can readily see that the presence of particles of one phase would interfere with the coalescence of particles of the other, but in the case of the single-phase system, we are confronted by the condition that grains of widely different size can remain side by side in apparent equilibrium, without being able to recognize any influence which would tend to maintain the higher surface energy of the smaller members.

The observations of the preceding paragraph will be recognized as merely an adaptation of Mr. Jeffries' views, carrying the assumption that a sufficiently long period of anneal would bring about a true equilibrium grain size throughout the entire region, superior to the present size in any localized region. Whether this is justified or not has no bearing on the immediate phenomena of ordinary anneal through a deformational gradient. I will confess to having pondered long over

[1] *Revue de Métallurgie*, vol. x, p. 725 (1913).

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this situation, particularly with respect to Professor Sauveur's micrographs, and the explanation offered by Mr. Jeffries is very acceptable.

In verbal discussion, Mr. Jeffries again quotes our assumption that a definite size of recrystallized grain corresponds to a given temperature of anneal by way of objection to the associated diagram of fragmentation and recrystallization. We have not ventured to suggest a quantitative treatment of recrystallization phenomena at large by means of this diagram. Aside from temperature, the factors of equilibrium as applied to grain size are partly obscure. The process is irreversible and leads to an apparent adjustment of grain size which may be of one type when the grains develop wholly within strained areas and of another type when the grains grow into previously unstrained areas (local deformation). Our diagrammatic treatment is intended to summarize the annealing effects after uniform reductions by rolling material of initial coarse grain as specified. Some modification would be necessary in applying this treatment to material of initially fine grain. The observations were made only on brass and bronze.

It is my opinion that an assumption of definite grain size for a given annealing temperature is valid in the case of brass even in the face of wide variation in the rate and period of heating. In brass laboratories, maximum temperatures of anneal are commonly predicted from examination of the metal without knowledge of the previous annealing conditions and I have knowledge of much accurate work of this sort. In the case of copper, similar observations are less satisfactory, perhaps owing to the influence of the rate-factor (Jeffries), or the presence of variable quantities of oxide.

In view of the fact that Mr. Jeffries states in our correspondence that he has conceived practically the same ideas as we have relative to the reasons for recrystallization, it does not appear desirable to leave the immediate subject of brass and enter into discussion of the other observations which he contributes. It is of great practical importance to characterize carefully the grain structures produced in different metals by heating the metal at different rates and annealing for different periods within the reasonable limits of mill practice. I am glad to express my appreciation of Mr. Jeffries' efforts in this direction.

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The Behavior of Stibnite in an Oxidizing Roast

BY H. O. HOFMAN,* BOSTON, MASS., AND JOHN BLATCHFORD,† OAK PARK, ILL.

(New York Meeting, February, 1916)

THE leading antimony mineral is stibnite. In smelting stibnite ore two processes are available, precipitation and roasting-reduction. The former is suited only for high-grade ores. As low-grade ores are more common than high-grade, roasting-reduction is of greater importance than precipitation. In the roasting process the aim may be to leave the oxidized antimony in the ore, or it may be to volatilize as much of the antimony as possible, collect the volatilized oxide as a rich intermediary product and smelt it for antimony, leaving the gangue poor enough to be considered a waste product.

Whichever way the roast is conducted certain difficulties inherent in stibnite are encountered. These are:

1. The low melting point of stibnite which, according to Pélabon,[1] is 550°C., according to Wagemann[2] 540°C., and according to Borgström[3] 546°C.

2. The ignition temperature: According to Friedrich,[4] stibnite, if heated in air, begins to oxidize at 290°C. if the size of a grain is 0.1 mm. in diameter; at 343°, if 0.1 to 0.2 mm.; and 430° if 0.2 mm.

3. The fusibility of a mixture of Sb_2S_3 and Sb_2O_3 which in the form of kermesite $(Sb_2S_3)_2.Sb_2O_3$ melts at 517°C.[5]

4. The volatility of Sb_2S_3 and Sb_2O_3, for which no numerical data appear to exist, although practical experience has shown that they are volatile at low temperatures.

As regards the oxidation of metallic antimony, we have the experimental evidence of C. F. Plattner[6] that when fused and brought to a red heat, it burns with a bluish-white flame to Sb_2O_3 which passes off as a

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[1] *Comptes rendus*, vol. cxxxvii, p. 920 (1903).
[2] *Metallurgie*, vol. ix, p. 518 (1912).
[3] *Chemical Abstracts*, vol. ix, p. 2364 (1915).
[4] *Metallurgie*, vol. vi, p. 169 (1909).
[5] Borgström. *Loc. cit.*
[6] *Die Metallurgischen Röstprocesse*, p. 162, Engelhardt, Freiberg, 1856.

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whitish fume, and that Sb_2O_3 can be further oxidized to Sb_2O_4 or Sb_2O_5. If Sb_2O_4 is considered to be antimonious antimonate, the compound falls in line with metallic antimonates which are formed if a metallic antimonide or sulphantimonide such as pyragyrite, tetrahedrite, or jamesonite, is subjected to an oxidizing roast—or if Sb_2O_3 in contact with a finely divided metallic oxide is heated in a current of air.

The chemical change which takes place in roasting stibnite is usually expressed by the equation

$$Sb_2S_3 + 9O = Sb_2O_3 + 3SO_2$$

but the Sb_2O_3 is further oxidized to Sb_2O_4. In carrying on a roast, the temperature of the ore is held at first at about 350°C., and the charge is rabbled more or less continuously to prevent or correct caking; later, when part of the Sb_2S_3 has been converted into Sb_2O_3 and Sb_2O_4, the temperature may be raised with advantage. Roasting changes the black sulphide into a yellowish-white oxide.

The aim of the present investigation was to study the changes stibnite undergoes when roasted at different temperatures; to determine the composition of the roasted product; and to ascertain the losses which occur.

The material used was crude stibnite of unknown origin from the metallurgical collection of the Massachusetts Institute of Technology. It was a compact grayish crystalline mass showing fine prismatic needles. The fresh fracture appeared silver-white and had a metallic luster. The general appearance was not unlike that of graphite. After grinding, it had a jet-black color. A chemical analysis gave 71.66 per cent. antimony and 0.96 per cent. insoluble. Calculating the antimony as the sulphide gives 100.3 per cent. Sb_2S_3; the excess over 100 is probably due to the presence of a small amount of metallic antimony.

The roasting furnace used was of the electric resistance type. It consisted of a porcelain tube, 10 in. long and 2¾ in. in diameter, wound with "nichrome" wire; this was surrounded by mineral wool, and the whole inclosed in a pipe of galvanized iron, 7½ in. in diameter. Temperatures were measured with a thermo-electric pyrometer.

The ore was ground to pass a 20-mesh sieve; a 5-gram sample was used, as this covered the bottom of the porcelain boat and filled it to about one-third of its depth.

During a roast the ore was frequently rabbled with a wire bent to the form of a hook. A piece of moistened litmus paper held above the surface of the ore served as a test for sulphur dioxide. Any odor, or change in the appearance of the ore was noted. The temperature of the furnace was raised gradually to the point at which it was desired to finish a roast, and held there until no more changes in appearance and weight of the charge occurred.

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Seven experiments were carried out, covering a range of finishing temperatures from 337.5 to 545.4°C. Experiment No. 3, in Table I, shows the manner in which the records were taken.

TABLE I.—*Record of Roast No. 3*

Time	Galvanometer, Deflection	Cold Junction, °C.	Temperature, °C.	Current, Amperes	Remarks
11 : 15	0.0	22.5	22.5	3.0	Start.
11 : 30	.....	22.5	.....	4.0	
11 : 45	.....	22.5	.....	4.0	
12 : 00	7.3	22.5	133.0	5.0	
12 : 15	11.0	22.5	175.0	5.0	
12 : 30	12.5	22.1	192.0	6.0	Faint odor of SO_2. Litmus begins to turn pink very slowly.
12 : 45	15.0	22.1	220.0	8.0	
1 : 00	18.0	22.5	257.0	10.0	Litmus changes to pink slowly.
1 : 15	24.0	22.5	328.0	10.2	Litmus changes to pink quickly; odor of SO_2 stronger.
1 : 30	23.2	22.5	315.0	9.0	Ore begins to look dull.
1 : 45	25.0	22.5	337.0	9.6	Ore is gray.
1 : 52	27.0	23.0	362.0	10.0	Very sharp odor of SO_2.
2 : 00	28.0	23.0	373.0	9.5	White ashes on rabbling, visible gas evolved.
2 : 15	27.5	23.0	368.0	9.6	Rabbling promotes formation of ashes.
2 : 30	28.0	23.5	374.0	9.6	
2 : 45	28.5	23.6	379.0	9.6	Odor of SO_2 not so strong; white kernels and gray powder form.
3 : 00	28.0	23.6	374.0	9.5	Litmus changes to pink readily.
3 : 15	28.0	23.8	374.0	9.4	
3 : 30	27.5	24.1	369.0	9.5	
3 : 45	27.0	24.5	364.0	9.8	
4 : 00	27.5	24.8	369.0	10.0	Litmus changes to pink very slowly.
4 : 15	28.0	24.8	375.0	10.0	Litmus changes to pink very slowly.
4 : 30	28.1	25.0	376.0	9.0	No change in litmus.
4 : 45	27.5	25.0	370.0	9.5	No odor of SO_2. Stopped.

	Grams
Weight of raw ore	4.9346
Weight of residue	4.4354
Loss weight	0.4992
Average finishing temperature	372°C.
Color of ground residue	Gray

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A summary of the observations made during the six roasts is given in Table II.

TABLE II.—*Summary of Observations in Roasting Experiments*

Temperature, °C.	Changes
196	Faint odor of SO_2.
219	Litmus begins to turn pink.
281	Ore becomes dull; odor of SO_2 very distinct.
336	White ashes form on rabbling, and some Sb_2O_3 is evolved.
380	Lowest temperature at which odor of SO_2 disappears.
400	Ore becomes sticky if temperature is raised quickly.

The first observation made in each roast was the evolution of sulphur dioxide; this was followed by a change in the color of the litmus paper at a slightly higher temperature. At 280°C. the ore began to lose its luster and became dull. Then followed a strong odor of sulphur dioxide which directly preceded a sudden visible formation of Sb_2O_3 at 336°C. The oxide forms spontaneously at this temperature in the form of little yellow grains which become white upon cooling. As the temperature is raised above 336°C., the grains seem to break up into a grayish powder. If the temperature is raised too quickly before all the sulphide has been changed into whitish grains of oxide, the charge becomes sticky; this is not the case if the temperature is raised gradually. In roasts Nos. 6 and 7 a small amount of Sb_2O_3 was deposited on the rim of the heating tube, but the temperature and time at which the deposit was formed were not noted. Tables III, IV, and V contain data on the roasting experiments and the results obtained.

TABLE III.—*Summary of Roasts*

Roast	Time, Hours	Raw Ore, Grams	Finishing Temperature, °C.	Loss of Weight, Grams	Loss of Weight, Per Cent.	Loss of Sb, Per Cent.
2	6¼	4.9994	337.5	0.3520	7.05	None
3	5½	4.9346	372.0	0.4992	10.10	None
4	3½	5.0480	407.5	0.6267	12.40	2.21
5	2¾	5.0382	454.5	0.7902*	15.70*	5.60*
6	3½	5.0028	493.5	0.5798	11.60	2.93
7	4¾	5.0388	545.4	0.5963	11.80	4.21

* This excessive loss in weight is due to some experimental accident which cannot be definitely placed.

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TABLE IV.—*Summary of Products*

Roast	Residue, Grams	Insoluble*		Sulphur, Per Cent.	Trivalent Antimony, Per Cent.	Total Antimony, Per Cent.	Sb_2S_3		Sb_2O_3		Sb_2O_4	
		Per Cent.	Grams				Per Cent.	Grams	Per Cent.	Grams	Per Cent.	Grams
2	4.6474	0.96	0.0446	13.46	77.22	76.67	46.9	2.1760	52.7	2.424	None	None
3	4.4354	0.96	0.0426	2.86	80.76	80.33	10.0	0.4439	87.7	3.905	None	None
4	4.4213	0.96	0.0424	None	75.00	80.01	None	None	84.0	3.713	12.7	0.561
5	4.2480	0.96	0.0408	None	61.85	80.23	None	None	52.7	2.216	46.1	1.978
6	4.4230	0.96	0.0425	None	52.44	78.70	None	None	31.4	1.389	66.5	2.942
7	4.4425	0.96	0.0427	None	41.77	77.85	None	None	6.8	0.303	91.5	4.061

* Based on analysis of raw ore.

TABLE V.—*Percentage of Antimony Compounds in Roasted Products*

Run	Total		Based on weight of raw ore		
	Per Cent.	Grams	Sb_2S_3, Per Cent.	Sb_2O_3 Per Cent.	Sb_2O_4 Per Cent.
2	100.00	4.6476	43.5	48.5	None
3	99.04	4.2915	9.0	79.2	None
4	97.63	4.3164	None	73.6	11.1
5	99.86	4.2348	None	43.8	39.2
6	98.91	4.3735	None	27.8	58.8
7	99.40	4.4170	None	6.0	84.5

A graphical representation of the results, given in Fig. 1, shows clearly the changes which stibnite undergoes when it is roasted. No change takes place until the gradual rise in temperature has reached 200°C. when SO_2 begins to be given off. The quick fall of the Sb_2S_3 curve shows that beginning with 200°C. a rapid oxidation takes place which is terminated at 400°C. It is fair to assume that the formation of Sb_2O_3 commences at the temperature when SO_2 is first detected. The Sb_2O_3 curve is therefore started at about 200°C; it has a rapid rise which corresponds to the quick fall of the Sb_2S_3 curve. Its highest point coincides on the temperature scale with the point at which the oxidation of Sb_2S_3 is complete; the Sb_2O_4 curve starts at the same temperature. As the temperature rises, the Sb_2O_3 curve falls and is accompanied by a corresponding rise in the curve for Sb_2O_4. From this it is seen that in roasting Sb_2S_3 there is formed at first only Sb_2O_3, and that this is converted into Sb_2O_4 only when all the Sb_2S_3 has been oxidized. Thus the amount of Sb_2O_4 formed is dependent upon the temperature and the time of the roast. It is assumed that there is no loss of weight below the temperature at which SO_2 begins to be evolved, *i.e.*, about 200°C. From this point on, the loss in weight increases with the temperature and the formation of Sb_2O_3; it reaches a maximum at about 450°C. then falls and afterward becomes

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nearly constant. The loss in antimony appears to be a function of the temperature; the curve shows that it increases with the finishing temperature with the exception of the jog at 450°C. which, as stated with Table III, is an unexplained experimental accident.

The following conclusions may be drawn from the investigation:

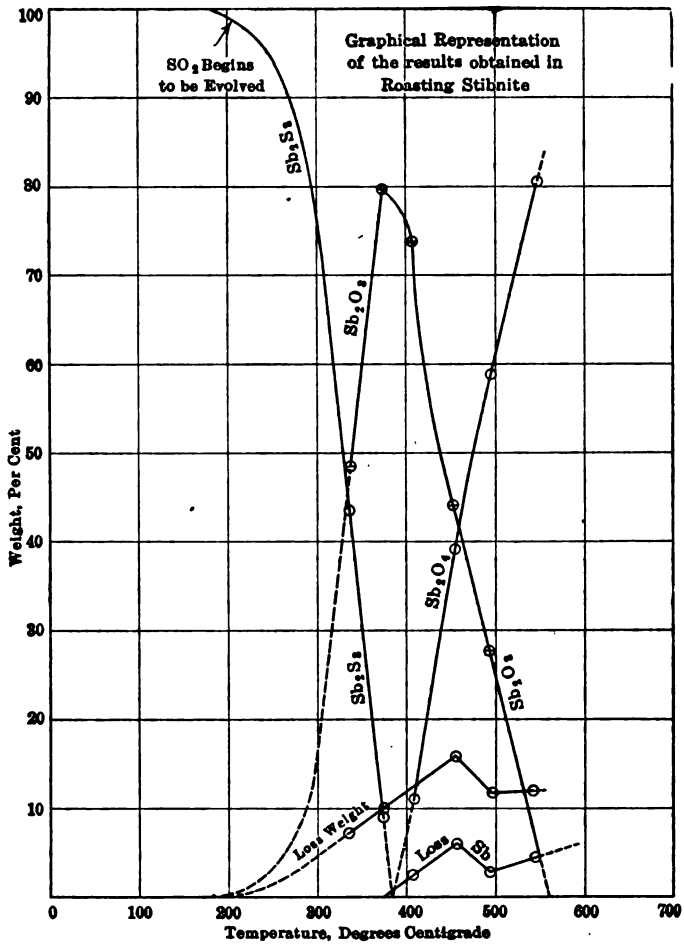

FIG. 1.—RESULTS OBTAINED IN ROASTING STIBNITE.

1. The ignition point of stibnite is approximately 200°C. as evidenced by the odor of SO_2 evolved and its action on litmus paper.

2. There is a sharp visible formation of Sb_2O_3 at 336°C. At this point white fumes are evolved.

3. The formation of Sb_2O_3 takes place slowly, probably according to the reaction:

$$Sb_2S_3 + 9O = Sb_2O_3 + 3SO_2$$

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4. It is possible to eliminate all the sulphur slightly below 400°C. without a large loss of antimony.

5. Stibnite is oxidized at first to antimony trioxide which begins to change to antimony tetroxide only when all the sulphide has been decomposed.

6. The amount of tetroxide increases as the trioxide decreases, the action being more rapid as the finishing temperature is raised.

7. The loss of antimony during the roast increases with the temperature.

A discussion of the chemical work is given in the paper by W. T. Hall and J. Blatchford, read at this meeting.

Discussion

Robert H. Richards, Boston, Mass.—I should like to ask Professor Hofman whether he is able to roast his stibnite without any comparatively large amount of loss of antimony?

H. O. Hofman, Boston, Mass.—Yes. The last statement I made was that the loss of antimony, during those tests, increased with the temperature, and that up to 372° C. there was no loss whatever in antimony, also that it increased to 4.21 per cent. with 545° C., the highest temperature that we obtained in our small furnace.

Robert H. Richards.—I should like to ask a second question, and that is, is it not true that stibnite can be roasted more completely than any other sulphide?

H. O. Hofman.—I always thought that pyrite was the mineral most easily roasted, and you could eliminate the sulphur more easily from it than from any other sulphide. We succeeded in our work in eliminating all the sulphur at a temperature slightly below 400° C.

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The Determination of Antimony in the Products Obtained by Roasting Stibnite

WILLIAM T. HALL,* BOSTON, MASS., AND JOHN BLATCHFORD,† OAK PARK, ILL.

(New York Meeting, February, 1916)

THE product obtained by roasting stibnite is likely to contain some unoxidized antimony trisulphide and a mixture of antimony trioxide and antimony tetroxide. It was desired to determine, as accurately as possible, the condition of the antimony as well as the total quantity present. Attempts were made to separate the trisulphide and the two oxides by methods based upon their varying solubilities in different solvents, but no satisfactory results were obtained in this way. It was found, however, that by determining the total antimony content, the antimony present as trioxide, and the antimony present as trisulphide, a good idea of the chemical composition of the roasted product could be obtained.

The total antimony was determined by dissolving the sample in concentrated hydrochloric acid, reducing the antimony entirely to the trivalent condition by means of hydriodic acid and eventually titrating the antimony back to the pentavalent condition by means of iodine in the presence of sodium bicarbonate.[1] The details of the procedure are as follows:

Weigh out 0.25 gram of the roasted product into a trapped flask (Fig. 1), add 4 grams of powdered tartaric acid, 2 grams of potassium iodide and 40 c.c. of concentrated hydrochloric acid. Boil the solution gently for 5 min. and then cool to room temperature by shaking the flask while holding it under running water. Then, very carefully discharge the iodine color by the cautious addition of 0.05-normal sodium thiosulphate solution, adding a little starch toward the last. Nearly neutralize the acid with ammonia solution, but leave the solution distinctly acid. Pour the slightly acid solution into 200 c.c. of water containing an excess of sodium bicarbonate and titrate to a permanent blue color with standard 0.1-normal iodine solution.

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† Analytical Chemist and Metallurgist.

[1] F. A. Gooch and H. W. Gruener: *American Journal of Science*, Series 3, vol. xlii, p. 213 (1891).

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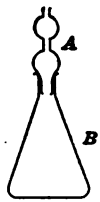
The determination of the unchanged antimony sulphide was accomplished by a method corresponding to that used in the determination of sulphur in steel.[2] The sample was dissolved in concentrated hydrochloric acid, the escaping hydrogen sulphide absorbed in an ammoniacal cadmium chloride solution and the sulphur in the precipitated cadmium sulphide determined iodometrically.

Inasmuch as all the antimony present in the roasted stibnite was soluble in concentrated hydrochloric acid, the solution remaining in the evolution flask after the determination of the sulphur could be used for the determination of the trivalent antimony. The details of the procedure are as follows:

To the solution remaining in the evolution flask after the determination of the sulphur, add 4 grams of powdered tartaric acid and carefully dilute with 100 c.c. of water. Carefully add 6-normal ammonia



A = Shortened $CaCl_2$ Tube
B = 400 C.C. Erlenmeyer Flask

FIG. 1.—APPARATUS USED IN DETERMINATION OF ANTIMONY IN ROASTED STIBNITE.

solution until the solution is only slightly acid and then pour the solution into a large beaker containing 5 grams of sodium bicarbonate in 200 c.c. of water. Titrate with 0.1-normal iodine solution as in the determination of the total antimony.

In computing the results, it is necessary to remember that antimony tetroxide may be regarded as antimonious antimonate:

$$2Sb_2O_4 = Sb_2O_3 + Sb_2O_5.$$

Since antimony pentoxide changes to the tetroxide when heated above 300°, it is fair to assume that each atom of pentavalent antimony found in the analysis corresponds to one molecule of antimony tetroxide. Then from the total quantity of trivalent antimony found, deductions must be made for the amount of tetroxide and for the trisulphide as found by the sulphur determination; the balance is the trivalent antimony corresponding to the quantity of antimony trioxide present.

Comparatively little information concerning the chemical properties of antimony tetroxide is to be found in the larger treatises on inorganic chemistry. From the literature, it was expected that considerable

[2] A. A. Blair: *The Chemical Analysis of Iron*, 7th Edition, p. 60. H Kinder: *Stahl und Eisen*, vol. xxviii, p. 249 (1908). Massenez: *Ibid*, vol. xxxii, p. 2089 (1912).

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difficulty would be encountered in dissolving antimony tetroxide in hydrochloric acid and it was surprising, therefore, to find no particular trouble from this source. Again, it was to be expected that in dissolving the roasted product, when some unchanged sulphide was present, some reduction of the pentavalent antimony and corresponding oxidation of the hydrogen sulphide to free sulphur would occur.

The accompanying table gives a summary of the results obtained in the analysis of the roasted stibnite.

Analyses of Roasted Stibnite

	Sulphur, Per Cent.	Trivalent Antimony, Per Cent.	Total Antimony, Per Cent.
Run II	.....	77.50*	76 60
	13.47	77.47*	76.74
	13.32	76.68	
Average	13.40	77.22	76.67
Run III	.....	80.79	81 25
	2.803	80.50	79 41
	2.918	80.98	
Average	2.860	80.76	80.33
Run IV	None	75.04	80.04
		74.96	79.97
Average	.....	75.00	80.01
Run V	None	62.01	80.41
		61.69	79.54
			80.04
Average	.....	61.85	80.23
Run VI	None	52.46	78.65
		52.41	78.74
Average	.....	52.44	78.70
Run VII	None	41.75	77.96
		41.79	77.73
Average	.....	41.77	77.85

* The values obtained when sulphide was present were a little high for trivalent antimony and a little low for sulphur.

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